A GEOLOGICAL EVALUATION OF THE TECHNICAL FEASIBILITY OF LONGWALL MINING IN THE GALT SEAM AT KIPP TRIAL MINE NUMBER 1787/E/01 (UP-P-137)

INTERIM REPORT

BY

PRECAMBRIAN SHIELD RESOURCES LIMITED

Work on this project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada.

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PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP

PREPARED FOR

PRECAMBRIAN SHIELD RESOURCES LIMITED

SUBMITTED BY

ASSOCIATED MINING CONSULTANTS LTD.

GOLDER ASSOCIATES

THE ASSOCIATION OF PROFESSIONAL ENGINEERS, GEOLOGISTS and GEOPHYSICISTS OF ALBERTA
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JANUARY 1983

AR-61A

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Precambrian Shield Resources Limited
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Attention: Mr. J.H.H. Chamberlin, P.Eng.
Vice President, Coal

Dear Sirs:

Re: Picture Butte Coal Project
Trial Mining Program - Kipp

We are pleased to submit our report **Picture Butte Coal Project Trial Mining Program - Kipp.** The Trial Mining Program (TMP) at Kipp has provided an abundance of data with regard to mining the Galt Seam. The major findings of the TMP at Kipp include the following:

- A coal parting left in the roof significantly improved roof conditions in general.
- Roadways driven east to west, west to east, and in a south-westerly direction parallel to the main cleat exhibited better roof conditions than those driven in a north to south direction.
- Test stations in 3 West and 4 West indicate that for the test period roof bolts alone (Rebar and Swellex) are capable of adequately supporting the roof.
- Instrumentation installed during the TMP suggests that roof extensometers installed during initial mine development will, after only a short period of time, give good indication of the longer term roof condition to be expected. The availability of early warning would indicate the potential for prediction and consequent control of strata conditions and behaviour. The Kipp roof behaviour, in our experience, is quite different from that in the Mountain/Foothills Mines where little or no warning of roof behaviour is provided to the operator. It would be of great benefit to develop this roof behaviour predictability further.

.....Page 2

We would like to express our thanks to Precambrian Shield Resources for the opportunity to provide our services and hope that we may be of further service to you in the future.

Respectfully submitted,

ASSOCIATED MINING CONSULTANTS LTD.

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PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

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PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SUMMARY OF FINDINGS, CONCLUSIONS AND RECOMMENDATIONS

The results of the Trial Mining Program in the Kipp Mine indicate the following:

a. Laboratory testing indicates that roof mudstones are stronger than the floor mudstones. No significant difference in strength was observed between mudstones and carbonaceous mudstones in the seam floor sequence.

b. The results of slake durability tests are consistent with results of previous laboratory testing on samples from Kipp and indicate that the floor mudstones will be susceptible to mechanical breakdown if exposed to alternating wet and dry conditions.

c. The high montmorillonite content present in the roof strata at Kipp explains the high swelling potential identified in free swell tests.

d. Roadways driven east to west, west to east, and in a southwesterly direction parallel to the main cleat exhibited better roof conditions than those driven in a north to south direction.

e. Roof conditions in 2 South were generally better than those found in 1 South. The east rib of 1 South sheared parallel to the rib line and dropped in the order of 30 cm from its mining horizon; shear fractures along the east rib were observed up to 60 cm above the roof line.

f. Test stations in 3 West and 4 West indicate that for the short term (months), roof bolts (Rebar and Swellex) are capable of supporting the roof.

g. Test Section I South indicates that over a very short term (weeks) roof sag along the east rib was accelerated with the removal of the roof to floor supports, and that roof bolts alone were inadequate. Roof collapse occurred in Test Section I South.

h. Floor heave in the development section is not excessive, at less than 5 cm for roadways 4.5 m and 5 m wide with 25 m and 30 m pillars of coal between roadways.

i. The mine was very dry during the TMP with the exception of some very limited water seepage in the vicinity of I South/5 West junction and the fault zone in 5 West.

j. Instrumentation installed during the TMP has shown that roof extensometers installed at the face during development will give a good indication of the longer term roof stability of the roadway in a short period of time. This indicates that adequate warning may well be available to control adverse strata conditions. This roof behaviour, in our experience, is quite different from that in Mountain/Foothills mines, where little or no warning is provided to the operator. It would be of great benefit to develop this predictability further.

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

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SECTION 1 - INTRODUCTION

1.1 INTRODUCTION

Associated Mining Consultants Ltd. with Golder Associates were retained by Precambrian Shield Resources Limited (PSRL) to undertake a Geotechnical/Strata Control Trial Mine Program in the Galt Seam at Petro-Canada's Kipp Mine and to provide preliminary mine design parameters for mining of the Galt Seam at Picture Butte (Figure 1-1).

The terms of reference for this study are outlined in a joint proposal submitted to Precambrian Shield Resources Limited by AMCL and Golder Associates, dated March 1982, as amended in April, 1982 and again in September 1982. The basic objectives of the program as these relate to the Kipp Mine were as follows:

- Evaluate available data from the Lethbridge area.
- Implement a geotechnical data collection program at Kipp Mine.
- Carry out a test mining and monitoring program at Kipp to test relevant geotechnical/strata control features.

The Galt Seam has previously been exploited in the Lethbridge area, the last mine closing in 1964. A literature study was carried out to document recorded mining conditions in this coalfield for comparison with those anticipated by previous geotechnical studies by Dames and Moore and Golder Associates in the Lethbridge area. The evaluation of previously accumulated data assisted in determination of geotechnical/strata control parameters at Kipp.

The geotechnical/strata control project incorporated underground studies at the Kipp Mine. The project schedule was such that the Trial Mining Program at Kipp took place during the latter part of 1982, and was completed on schedule in December, 1982. This report includes a description of the TMP along with the results of instrumentation and monitoring, and the results of geotechnical drilling and testing.

1.2 REPORTS AND DOCUMENTATION

1953 - 1971 F. Grant, et al. (T1)
Untitled: Seven papers on roofbolting, subsidence, ground stress in Lethbridge and Canadian mines.

1979 Dames & Moore (R14)
Report on Geotechnical Assessment for Shaft, Lethbridge Coalfield, Lethbridge, Alberta.

1979A Norwest (R17)
1979 Data Package for the Lethbridge Coal Project.

1979B Norwest (R18)
Maps to Accompany 1979 Data Package for the Lethbridge Coal Project.

1980A Robinson, Dames & Moore (R22)
Report on Subsidence Study, Lethbridge Coal Project, Lethbridge Alberta.

1980B Robinson, Dames & Moore (R26)
Review of Groundwater Hydrology Conditions, Lethbridge Coal Project, Lethbridge, Alberta.

1980 Petro-Canada (R27)
Untitled Report: Field operations and structural geology - Site Report.

1982 Petro-Canada
Application to ERCB for TMP - including Appendix I of letter to D. Henderson dated July 5, 1982 showing anticipated seam section, anticipated quality of run-of-mine coal and copy of Drawing No. 6/2 showing support of Intersections.

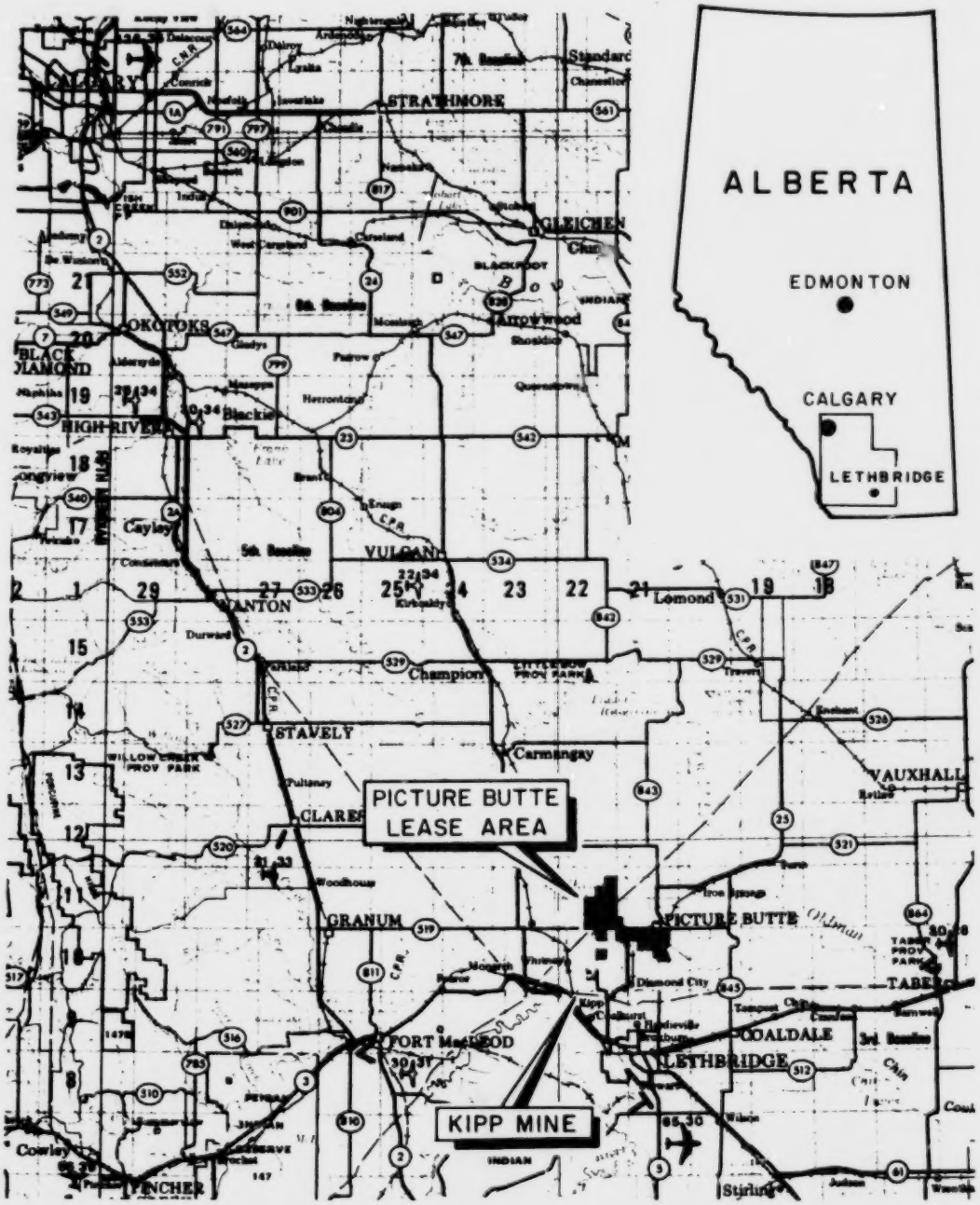

FIGURE 1-1

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

LOCATION MAP

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

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SECTION 2 - DESCRIPTION OF THE KIPP MINE AREA

2.1 LOCATION

The Kipp property is located within the Lethbridge Coalfield, some 15 km north of Lethbridge, Alberta.

Numerous small to medium-sized underground mines have operated in the Lethbridge area with annual production peaking at around 700 000 t in 1913. The larger mines in the area were operated by Lethbridge Collieries Ltd., a CPR subsidiary, and production was primarily for domestic uses with some significant railroad consumption in earlier years. Coal production has been exclusively from the Galt Seam which is a persistent unit throughout the area. The Shaughnessy or Galt #10 Mine was the last major mining operation in the area, closing in 1964.

2.2 TOPOGRAPHY

The topography of the area consists of a gently undulating plain. The area is primarily an agricultural region and is well developed and serviced by a network of surfaced roads including Highways #3 and #4 and the Crowsnest Branch of CPR which runs through the south end of the Kipp leases. The area is open and relatively flat at an elevation of about 900 m above sea level with the exception of the St. Mary and Oldman Rivers, which are incised 60 m to 90 m below the prairie level.

2.3 GENERAL GEOLOGY

The Lethbridge Coalfield is located on the east flank of the Alberta Syncline. A large part of the area is underlain by continental sediments

of the Upper Cretaceous Belly River Group (Figure 2-1). Coal seams occur within the Lethbridge Zone which lies in the uppermost 25 m to 30 m of this group. The Lethbridge Coal Zone or Whole Seam (at Kipp) contains a number of coal seams of which one seam is persistent and is correlated as The Galt Seam. The regional dip is to the west.

The comments which follow pertain only to the geology of the Kipp Mine area, based on information from previous studies and from data accumulated during the TMP.

2.3.1 Recent Overburden

Glacial till material predominates over the Lethbridge Region. At Kipp, glacial deposits extend to a depth of approximately 59 m (Dames and Moore, 1979), and consists of clays and silts with frequent sand and gravel layers. The till is underlain by some 11 m of sandy gravels, which in turn are underlain by bedrock.

2.3.2 Bedrock Stratigraphy

Strata in the Kipp area are generally flat lying. Bedrock consists of massive mudstone of the Bearpaw Formation underlain by a sequence of laminated sandstone, siltstone, mudstone and coal seams of the Oldman Formation of the Upper Cretaceous Belly River Group. The upper part of the Oldman Formation contains numerous coal seams. The Galt Seam is the only seam of economic interest, and occurs near the top of the Oldman Formation.

The seam section and immediate roof and floor sequence is shown in Figure 2-2. The main geologic units within this sequence are as follows:

- Main Roof
- Main Rider Seam
- Immediate Seam Roof
- Galt Seam
- Floor Strata

a. Main Roof

The Main Roof, consisting of all strata above the Main Rider Seam, comprise the uppermost Oldman Strata and the massive mudstone of the Bearpaw Formation.

b. Main Rider Seam

The Main Rider Seam, occurring throughout the area, occurs near the top of the Oldman Formation and comprises two coal bands, the Upper and Lower Riders separated by a mudstone parting. In the mine area the Lower Rider consists of argillaceous coal and is approximately 0.3 m thick. The Upper Rider is between 0.4 m and 0.5 m thick.

c. Immediate Seam Roof

The Immediate Seam Roof comprising strata between the Galt Seam and the Main Rider in descending order of lithologic sequence is as follows:

- Main Rider
- mudstone
- siltstone
- carbonaceous mudstone/coal band
- silty mudstone ("Ostrea Shell Zone" equivalent)
- mudstone
- Galt Rider (argillaceous coal)
- mudstone
- Galt Seam

The total thickness of the section from the Galt Seam to the bottom of the Main Rider Seam (Upper Rider) at Kipp is approximately 7 m. The relative position of the different units within the sequence is generally constant and the only variables occur as facies changes.

The thickness of the mudstone band immediately above the Galt Seam generally decreases in a northerly direction across the area from a maximum of approximately 0.7 m in the southwest to approximately 0.3 m in the northwest. In the mine this mudstone band is 0.35 m thick.

The Galt Rider which consists of coal to argillaceous coal is a constant feature across the Kipp area and in the mine has a thickness of 0.18 m.

The Ostrea Shell Zone does not exist as a shell bed at Kipp but is represented stratigraphically by a compact cemented silty mudstone band with arenaceous laminae.

A carbonaceous mudstone to argillaceous coal band exists approximately 1.8 m to 2 m above the top of the Galt Seam, and was observed in the roof sequence exposed in the mine (1 South, roof fall area). Overlying this unit is a 0.25 m thick, hard, competent, dark grey siltstone band, with occasional fine sandy laminae.

d. The Galt Seam

The Galt Seam at Kipp consists of three coal bands split by two shale partings (Figure 2-2). The total thickness of the Galt Seam as encountered in the mine is 1.65 m.

e. Floor Strata

The floor strata immediately below the Galt Seam consists of mudstone and carbonaceous mudstone. This strata grades into an interlayered sequence of mudstones, siltstones and true coal bands. Bentonitic bands are located within 2 m of the base of the Galt Seam. Further down the sequence the lithology grades into wavy bedded, arenaceous siltstones and sandstones.

2.3.3 Structure

The main structural feature at Kipp is the pronounced bedding. Jointing as systematic fracture sets have not been observed in the roof or floor strata, except near faults.

The coal, both the Galt Seam and the Galt Rider, exhibit a pronounced cleat oriented at an azimuth of 060°, dipping nearly vertical. A poorly developed secondary cleat lies at right angles to the primary cleat.

The limited development within the Kipp Mine intersected two fault zones, one in the northwest, and the other in the southeast extremeties of the mine (Figure 3-1). The fault zone in 1 West is a 10 m wide zone including minor fault surfaces striking north-easterly and north-westerly, dipping 65° to 35° predominantly northwest and southwest, with a few minor surfaces dipping to the southeast. The effect of this fault zone has been to displace the

western block downward at least 4 m. The other fault zone was encountered at the end of the TMP in 5 West. The main fault surface strikes 073° and dips southerly 65° near the roof to 35° near the floor. Minor structures in the hanging wall block indicate that this fault is a normal fault, south block down (Figure 2-3). Limited probing with an auger drill indicated that the stratigraphic throw could exceed 10 m.

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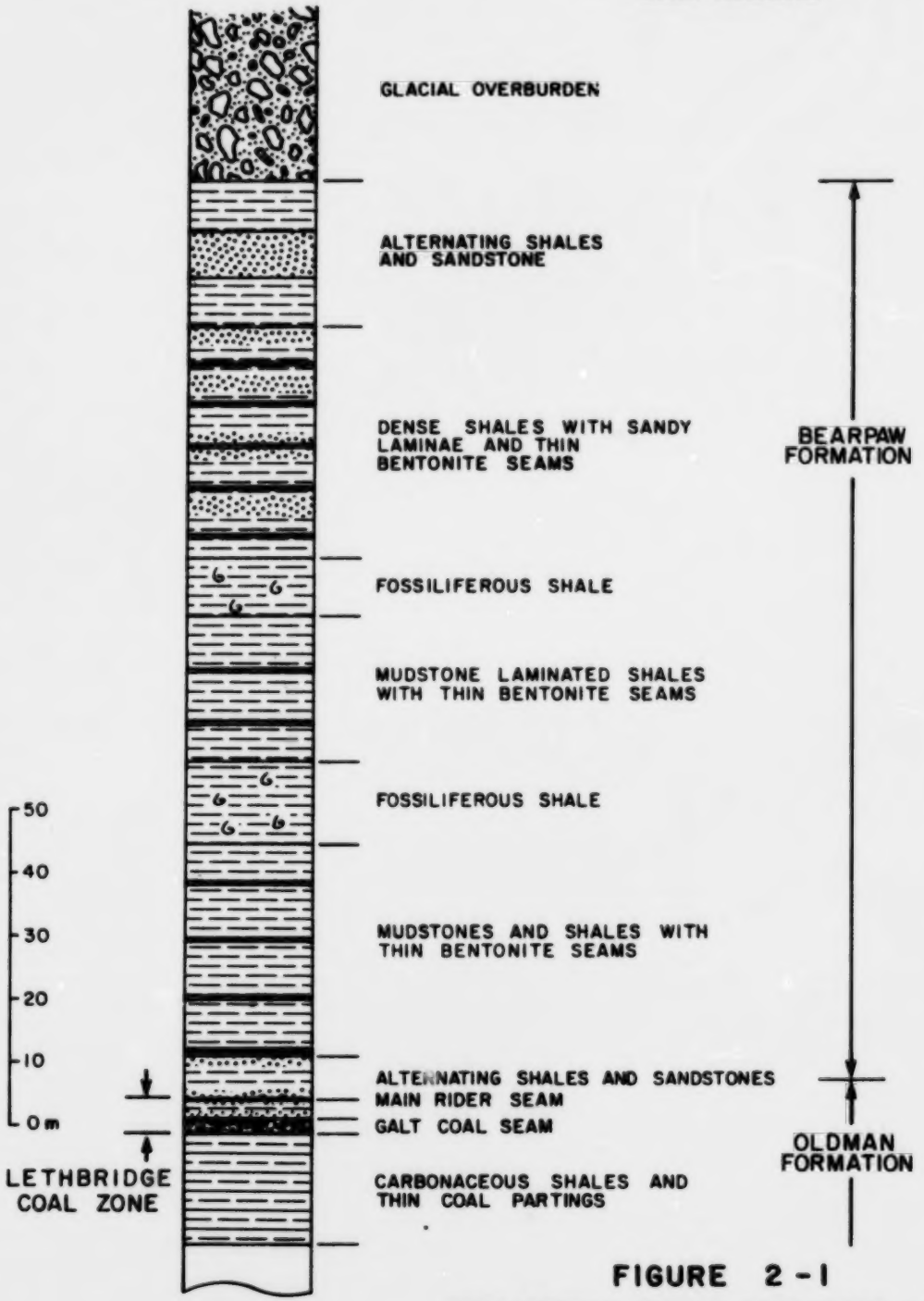

FIGURE 2-1

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

GENERALIZED STRATIGRAPHIC SECTION
LETHBRIDGE AREA

PA73-021

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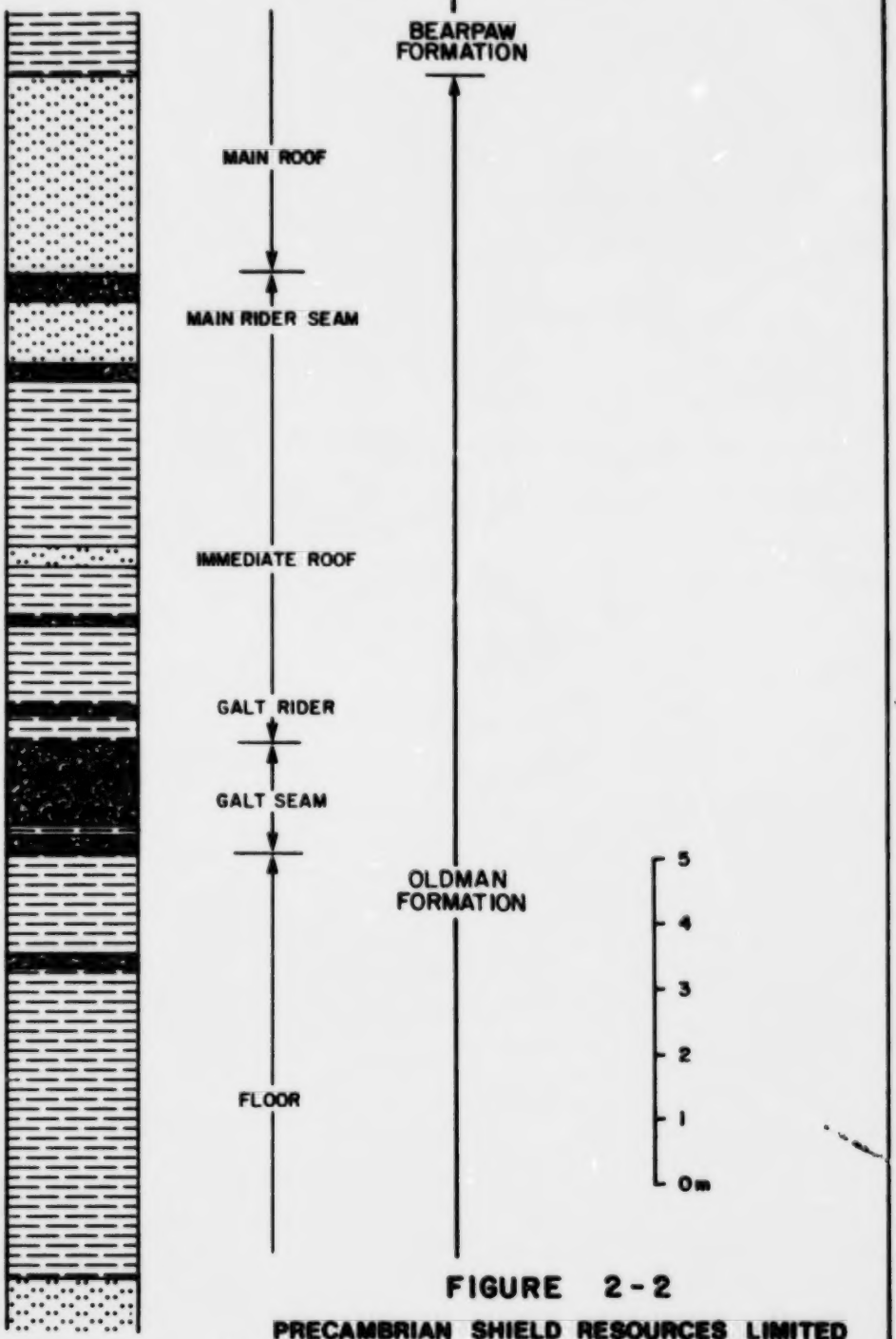

FIGURE 2-2

PRECAMBRIAN SHIELD RESOURCES LIMITED

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PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

GEOLOGIC SECTION

KIPP

COAL

SILTSTONE

MUDSTONE

PA75-022

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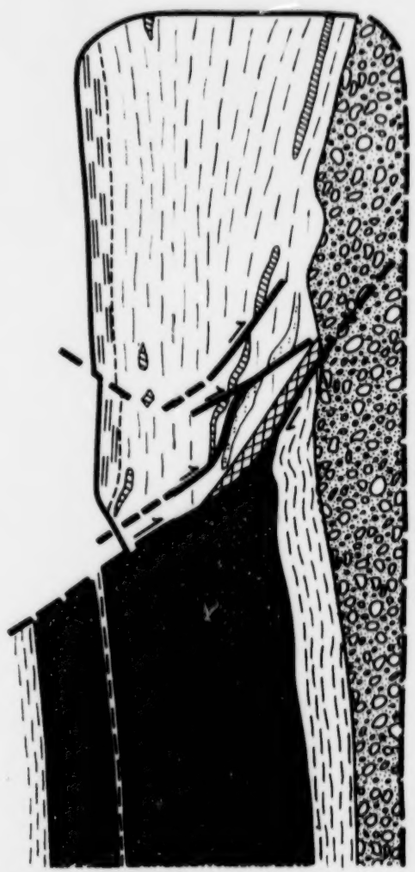
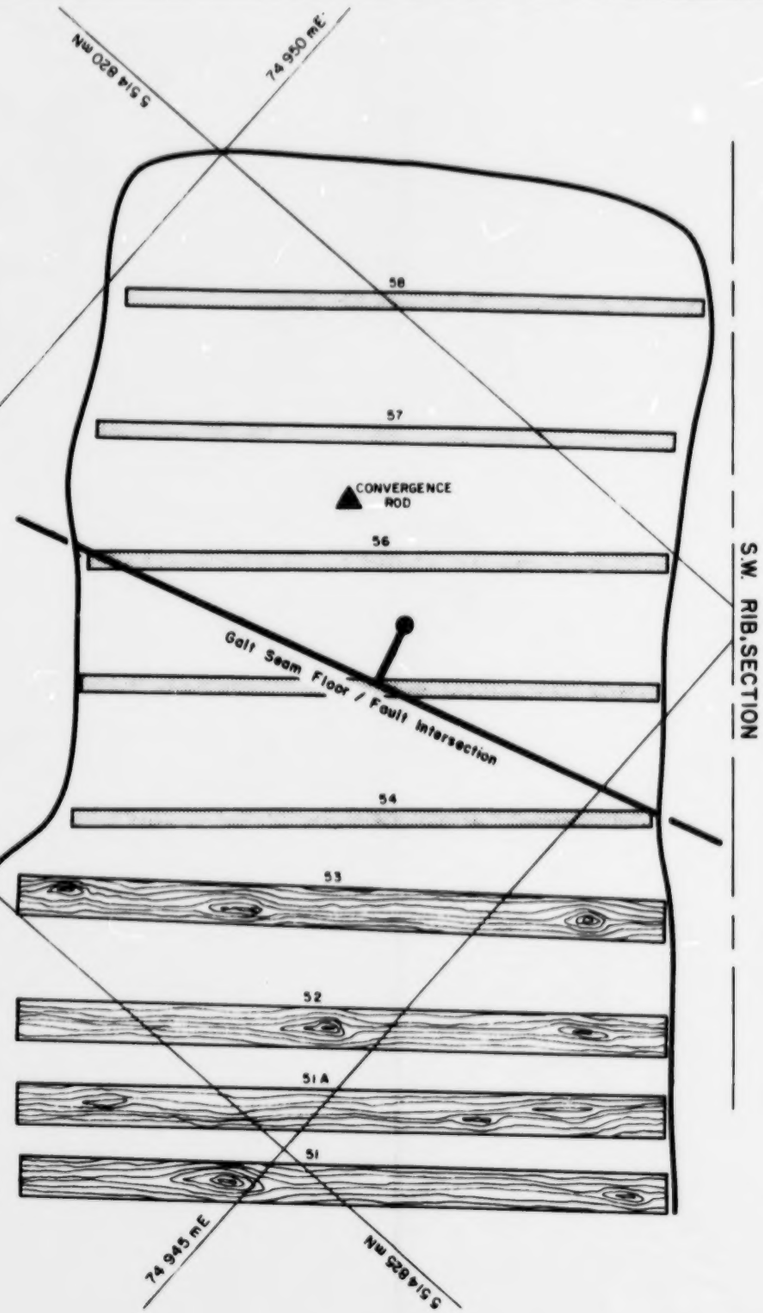
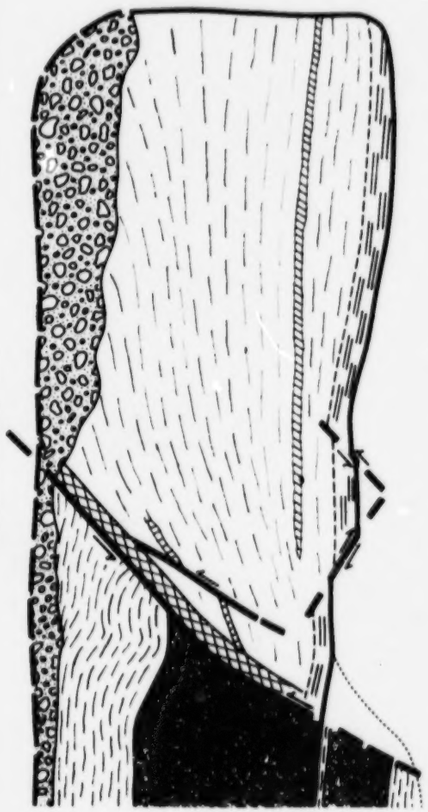


N.E. RIB SECTION



S.W. RIB SECTION



LEGEND

- Shear Zone
- Coal
- Carbonaceous Mudstone / Shale
- Mudstone / Shale
- Ironstone
- Ferruginous Siltstone
- Rubble

0 1 2 3 4 m

3 - T.M. GRID COORDINATE SYSTEM
GRID REF. DWG No. K83-116-A

FIGURE 2-3

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

FAULT IN 5 WEST

PA73-047

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SECTION 3 - TRIAL MINING PROGRAM - KIPP MINE

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December 7/1982

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SECTION 3 - TRIAL MINING PROGRAM - KIPP MINE

3.1 INTRODUCTION

The TMP at the Kipp Mine was started on July 26, 1982 and was continued until October 27, 1982. The overall mine plan (Figure 3-1) shows the mining carried out by Petro-Canada during 1980-81, where mining was started in the TMP, and the location of geotechnical drillholes. A total of 401 m of roadway was driven in the TMP. Figure 3-1 also shows both fault zones encountered.

3.2 DESCRIPTION OF MINING METHODS

3.2.1 Coal Cutting and Transport

All headings were driven with a single mining crew with a Joy Model 12CM Continuous Miner. The cutting head on the miner was 2.7 m wide, but 5 m wide headings were cut in order to provide manoeuvering room for the miner in negotiating 90° intersections. Blasting was used to aid in starting intersections since the continuous miner had too large a turning radius to make the required cuts.

Development faces were cut in either two or three passes. The initial advance along 1 South indicated roof slabbing along the east rib line. Different sequences of face cutting were tried in an attempt to overcome this situation, as follows:

- Two cuts - east rib followed by west rib
  west rib followed by east rib
- Three cuts - a centre cut followed by a trim cut along the west and the east ribs or vice versa

All these combinations were tried; but no single sequence alleviated the roof condition. The cutting sequence was therefore left to the discretion of each underground foreman to cope with locally encountered conditions.

Coal was transported from the face by a Joy 10-SC Shuttlecar, then by belt conveyor to the shaft and hoisted to the surface in a 3 t bucket.

3.2.2 Mine Plan

The original application for the TMP was made May 13, 1982, as a revision to the existing licence for the Kipp Mine (Mine No. 1787/E/01, Permit C 79-19A, Licence No. 80-23).

The initial proposed plan (underground layout and proposed roadway alignment), is shown in Figure 3-2A. As indicated on the plan, 360 m of roadway are oriented along the cleat, 136 m of roadway at 90° to the cleat and 94 m of roadway in the north/south orientation.

Between July 28 and August 13, 1 South and 3 West were driven to a point where 2 South could intersect 3 West. Observations of general roof conditions at this time indicated a much more stable roof in 3 West than in 1 South. The roof in 1 South was found to break away prior to support installation and girders deflected

several centimetres after installation. None of these features were encountered while driving 3 West.

Since 60% of the roadway driveage in the proposed plan was to be parallel to the cleat, as in 3 West, AMCL/GA recommended PSRL modify the plan to include more north/south driveage. This revised plan would enable testing to be done under the more difficult conditions paralleling the township grid system and, therefore, generate more meaningful data. The second plan (Figure 3-2B) was submitted to the ERCB August 19, 1982. This second plan involved extending the two original north/south roadways on 40 m centres. The southern extension of these roadways would then turn at 90° to the cleat angle. This second plan afforded an additional 192 m of north/south roadway which was not included in the first plan.

On October 8, 1982 a third revised plan was submitted to the ERCB. The intent of this third plan was to obtain information on a roadway driven at 90° to the cleat. The third plan (Figure 3-2C) proposed an easterly extension to 5 West which would be turned 90° to the cleat. Information on caving characteristics was to be obtained in this heading by widening out on retreat. The first part of this revised plan was followed but 5 West intersected the fault and the caving attempt was abandoned.

3.2.3 Roof Support

Three basic types of roof support were used in accordance with the Mine Manager's Support Rules (Figure 3-3), namely steel H-beams or wood booms, and roof bolts and straps.

Three types of H-beam sections were installed as roof support. These were:

Letter Designation	Description of Section	Section Modulus
A	8" x 8" Double H-beam (20 cm x 20 cm)	Unknown
B	8" x 6" x 35 lb/ft H-beam (20 cm x 15 cm x 52 kg/m)	28.77 in^3 (471.5 cm^3)
C	6" x 5" x 25 lb/ft H-beam (15 cm x 13 cm x 37 kg/m)	14.56 in^3 (238.6 cm^3)
D	8" x 12" x 70 lb/ft Double H-beam (20 cm x 30 cm x 104 kg/m)	57.54 m^3 (942.9 cm^3)

Sections A and B were installed in 4.6 m lengths and Section C was installed in 4.6 m and 4.1 m lengths, depending on the width of the heading being driven. Section D was fabricated in lengths of 6.1 m to be used as bearing girders at junctions.

The steel designated by the letter A listed above comprised two 8" x 4" H-beams welded to form a box section, the same configuration used by Petro-Canada in the original Petro-Canada mining trials. Section A was used for the first 11 girders installed in 1 South. However, it was found that the roof load encountered caused excessive deflection, requiring the replacement of one beam and the addition of more girders.

Steel Section B, used for the majority of the test program, was a rolled steel joist (RSJ) supplied by Dosco containing more metal in the web to prevent web failure.

A third section, Section C, introduced in the last week of September was used in 2 South, 5 West and 1 South. The section

modulus and thus load bearing capacity of Section C was approximately 50% that of Section B.

Section D was fabricated from two 8" x 6" x 35 lb/ft girders welded together in lengths of 6.1 m. This section was used for bearing girders at intersections.

To provide additional information of roof supports, wood booms were tested in selected headings. These booms were of greater than 15 cm diameter and were dapped at both ends to provide flat surfaces for setting posts.

The two types of roof support bolts tested were Rebar roof bolts and Swellex roof bolts. Both types of roof bolts were installed in 1.83 m and 2.13 m lengths in selected test areas. Physical characteristics of both types of roof bolts are listed in Table 3-1.

Roof bolts spaced four abreast on a 1.2 m x 1.2 m pattern were installed between the sets. Two steel straps were placed parallel to the sets each secured by two roof bolts (Figure 3-3). Straps measured 1.8 m x 0.3 m x 3 mm thick with two holes drilled on 1.2 m centres for roof bolts. All roof bolt holes were drilled with stopers.

Full-column resin-grouted bolts, subsequently referred to as Rebar bolts, consisted of 20 mm diameter grade 400 rebar with a 29 mm forged head in conjunction with a 10 mm x 10 mm x 6 mm plate domed washer. 1.83 m long bolts were fully grouted using 6 resin cartridges. Initially 6 resin cartridges with a set time of 5 to 9 minutes were used per bolt resulting in delays in the roof bolt installation. This was excessive since it prevented the use of the stoper while the bolt was setting. After August 13, starting in 2 South, a quick set cartridge (stated setting time 1 minute) was

placed at the top of the hole. Setting times of 2 to 4 minutes were recorded for this procedure.

Swellex bolts were installed in all headings except 3 West and 2 South. The Swellex system involves a reshaped hollow metal tube which conforms to the diameter of the hole when pressurized with water. The water pressure within the tube is released when the high pressure water source is disconnected. The pneumatic water booster pump is air-powered and can be connected to the mine water supply or operated with an independant water reservoir. Bolts are installed with a chuck switch mounted on an installation arm which allows the face worker to pressurize bolts while standing under secured roof.

Observations on bolt failures were made in Test Section I South along the edge of the caved section. In the south end of the caved area two Swellex bolts were recovered which failed in tension 20 cm and 70 cm from the plate. There was some elongation in the tube which decreased the tube diameter and wall thickness before failure. Rebar bolts exposed in the north boundary of the cave were not recovered, however one bolt which was exposed in the fall appeared to have the plastic cartridge casings wrapped next to the bolt over a 0.5 m section, interfering with proper bonding of steel to resin to rock.

3.2.4 Mining Horizon

Several selected mining horizons were mined to provide information on roof stability. Longitudinal-sections of the horizon changes are shown in Figures 3-4 and 3-5.

Mining completed by Petro-Canada prior to the TMP was carried out by leaving a coal roof comprising the Galt Rider Seam. This

roof was difficult to maintain and a new horizon was established at commencement of the TMP.

The roof of 1 South was established at a thin shale parting 40 cm below the top of the Galt Seam. The immediate drop in the roof horizon required floor brushing between Sets #95 and #99 in the old Petro-Canada workings.

Initial driveage in 2 South maintained a level roof horizon while gently lowering the floor from Set #4 to Set #12. The roof horizon was changed between Set #12 and Set #13, from the mudstone bed between the Galt Seam and Galt Rider Seam, to the shale parting 40 cm below the top of the Galt Seam.

An experimental horizon change was tested in 2 South between 4 West and 5 West intersections (Set #83 to #86). The interval between the mudstone bed above the Galt Seam and 30 cm to 50 cm of mudstone beneath the Galt Seam was mined. This horizon was mined for the remainder of 2 South and continued to Set #15 in 5 West after which the original mudstone band in the Galt Seam was used as roof.

An horizon change was made in 5 West after Set #36 in an attempt to form a roof line within the uppermost 20 cm of the Galt Seam. Since there were no easily identifiable markers for the miner operator to judge coal thickness, the coal veneer varied up to 30 cm. As driveage approached the fault this horizon became more difficult to maintain and the roof required blocking above the sets to prevent further caving.

3.2.5 Support Changes

The floor brushing procedure in 1 South (Petro-Canada Set #95 to Set #99) left the base of the legs standing on a berm. Both legs

for each girder were replaced with longer posts to prevent the legs from being 'kicked in' off the berm towards the centre of the roadway.

Old wood booms were initially installed for Sets #41 to #51. Due to the poor condition of the wood, and excessive bending and cracking, 8" x 6" H-beams were installed as replacements within 4 days. Sets #54 to #61 were installed with new wood booms, however, they too required replacement with 8" x 6 H-beams within 4 days, due to cracking. Sets #52 and #53 were steel H-beams originally.

At the start of mining in 2 South the legs of only 2 sets (#31 and #32 Petro-Canada) were replaced.

Major roof support changes were made in 2 South (extension) between 4 West and 5 West intersections. Initially wood sets were installed from Set #82 to #104 inclusive. A mining horizon change was made from the shale band in the Galt Seam (Set #82) to a parting within the mudstone band below the Galt Rider Seam (Set #86). Due to the combination of wood booms with a lower section modulus than the 8" x 6" girders and the change in mining horizon roof pressure caused excessive deflection, cracking and breakage of some booms. On September 21, after the installation of Set #97, the breakage of booms required the centre-posting of Sets #82 to #86 under the lip created by the mining horizon change. The last wood boom was installed on September 23 and by September 27 all wood booms had been replaced with 6" x 5" girders except Sets #82 and #99.

The final change made in 2 South was the removal of alternate 8" x 6" girders, between Sets #16 and #32 inclusive. Since roof

conditions in this section of roadway were good, alternate girders were replaced with wood sets without roof deterioration.

Four additional wood sets, #17A, #18A, #19A and #20A were installed in 5 West to aid in supporting the lip created by the mining horizon change. The new sets were placed when cracks in the existing booms were observed.

The last mining horizon change was made in 5 West between Sets #36 and #37. Though no problems with existing support were recorded, two additional wood sets, #35A and #36A were installed to provide additional support. The supports were installed during a lull in mining activity when the fault was intersected in 5 West. Apparently the two sets were erected as a safety measure rather than an emergency repair.

Another wood set was erected between Set #51 and #52 to provide additional support for the heavily blocked roof encountered several metres prior to the fault intersection. This set (#51A) was installed at the same time as Sets #35A and #36A.

Instrumentation test stations were installed in 3 West, 1 South and 4 West. Sets were removed over the entire test section interval but bolts and straps were left in place. Listed below are the sets removed for each of the test sections:

Roadway	Date	Set #	Type of Set
3 West	Sep. 21/22	8 to 22	8" x 6" Girder
1 South	Oct. 13	109 to 114 115 to 125	8" x 6" Girder 6" x 5" Girder
4 West	Oct. 25	10 to 25	Wood

The final position of centre-posts are marked in Figure 3-5. Double centre-posts were installed adjacent to both ends of each test area during the removal of the sets. Centre-posts were also erected in the vicinity of the electrical switchgear in 2 South near the intersection of 3 West to provide additional roof support and for fencing purposes.

Centre-posting was later installed along 1 South to 4 West. Since the belt conveyor, continuous miner and shuttlecar were taken out of the mine at the end of the TMP, centre-posting along 1 South was removed to facilitate equipment removal and then reinstalled. Minor leg repairs are detailed on Table 3-2.

3.3 DISCUSSION

During the course of the TMP it was found that roof stability was noticeably improved when driving in an east-west or south-westerly direction as opposed to a north-south direction. This is illustrated in Figure 3-6, which shows considerable sloughing along 1 South and portions of 2 South when driving in a southerly direction. It should be noted that virtually no roof sloughing took place when driving north in 1 South from Sets #101 to #131. The small roof fall which did occur, near breakthrough, can be attributed to questionable operating procedure of leaving roadway unsupported over much of one shift.

Good roof conditions were encountered in 3 West, the only roadway driven along the cleat. It was noted that sloughing of coal from the rib was more than in north-south roadways, but this was to be expected as the cleat fracture planes were parallel to the axis of driveage. Good roof conditions were also encountered in 4 West which was driven west.

5 West was driven east between Sets #1 and #42 and was turned perpendicular to the cleat for the remainder. The only roof problems

encountered in 5 West were in the vicinity of the fault where up to one metre of blocking was required.

From the point of view of roof stability both types of bolts, Swellex and Rebar, worked satisfactorily. The only installation concern was that holes were drilled up to 20° off vertical to make room for the stoper and steel, in local low roof conditions. The Swellex system of bolting was found to be procedurally more efficient than Rebar bolts in that cartridges did not have to be handled and the stoper was not tied up waiting for the resin to set. In addition, the Swellex bolts were lighter and were installed into the holes with no resisting pressure as was the case with the Rebar bolt when penetrating bags of resin. The disadvantage of the Swellex system lay with the design of the auxiliary equipment comprising high pressure pump and installation tool. On two occasions the bolting operations were stalled when the 'O' ring was blown from the installation tool, requiring the services of a mechanic for repair. This equipment should be made more robust for underground use.

In general the Swellex bolting system was preferred by the face workers, however, it is recommended that resin bolts and cartridges be kept at the face should failure with the Swellex pump or installation tool occur.

Most heavy roof conditions can be attributed to either insufficient beam strength or a change in mining horizon. In 1 South roof sloughing was immediately encountered at the start of driveage due in part to the light section steel 8" x 8" double H-beam installed as Sets #1 to #11 during a change in the mining horizon. When the 8" x 6" girders were used for supporting 2 South sloughing was not encountered though the better roof conditions may be attributed to lower roof pressure.

Wood Sets #41 to #61 in 1 South did not have sufficient section modulus to withstand the forces exerted by the roof. Assuming tensile strength of 6900 kPa (1000 psi) for wood, a wood boom 23 cm in diameter at the centre can support only 5% the load of an 8" x 6" x 35 lb/ft girder and only 10% the load of a 6" x 5" x 25 lb/ft girder.

Use of wood sets in 2 South from Set #82 to #104 also created the same problems such as excessive deflection and cracking encountered with wood sets in 1 South. The problem was further accentuated by the change in the mining horizon to the mudstone band between the Galt Rider and Galt Seam. As a result most wooden beams were replaced with steel.

Wood sets and roof bolts are satisfactory if roof conditions are good (ie: no roof spalling) such as those encountered in 4 West, 3 West and in 5 West between 1 South and 2 South. It appears that if the roof strata is unbroken, it acts as a beam spanning the roadway and resting on the coal rib pillars, resulting in less load on the installed support sets. Roof bolts increase the strength of the roof strata beam. However, even with roof bolts, if the roof is in poor condition such as heavily cracked, or spalled along one side or the other, then the installed sets must carry an ever increasing weight of broken material above the roof to the point of supporting the entire weight of the broken roof material when the strata beam fails.

The mining development sequence is illustrated by the bar chart in Figure 3-7. As indicated in the lower half of the chart, more delays affecting mining advance can be attributed to support changes and repair than any other combination of causes. Of the support changes, most were made in 1 South and 2 South.

TABLE 3-1

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP

LIST OF ROOF BOLT INFORMATION

Swellex Bolts

Installed Length	1.8 m (6') 2.1 m (7')
Unexpanded Outer Diameter	25.5 mm (0.93")
Maximum Outer Diameter	41 mm (1.51")
Wall Thickness	2 mm (5/64")
Fracture Load	100 kN (11.24 tons*)
Water Pressure	30 MPa (4350 psi)
Recommended Hole Diameter	38 mm (1½")
Manufacturer	Atlas Copco

Grouted Rebar Bolts

Installed Length	1.8 m (6') 2.1 m (7')
Nominal Diameter	20 mm (0.79")
Yield Strength	400 MPa (58 000 psi)
Fracture Load	126 kN (14.2 tons*)
Manufacturer	Stelco
Recommended Hole Diameter	38 mm (1½")
Resin Type	Celtite HU0204 (2-4 Min.Set Time @ 72°F) Celtite M50 (50 sec. Set Time @ 72° F)
No. of Cartridges per Hole (2.1 & 1.8 m holes)	1-M50 @ top of hole 5-HU0204

* Short tons

TABLE 3-2
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

MINOR LEG REPAIRS

Roadway	Set #	Repair	Date	Comment
#1 South	7 and 8	East legs replaced	Jul 30	Legs not set properly by mining crew
	9, 19 and 20	East legs replaced	Aug 4	No reason stated, likely improperly set
	7, 8 and 12	West legs replaced	Aug 6	Water on floor from broken water line caused posts to sink into floor
	Bearing Girder at #1 West	Both legs replaced	Sep 8	Bearing girder was twisting due to improper installation
	Bearing Girder at #4 West	Both legs replaced	Sep 29	No reason stated, likely knocked out by shuttlecar
#2 South (extension)	Bearing Girder	East leg at #3 West replaced	Aug 25	Post blasted out
	Bearing Girder at #5 West	Both legs replaced	Sep 29	No reason stated, likely damaged by shuttlecar

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Golder Associates

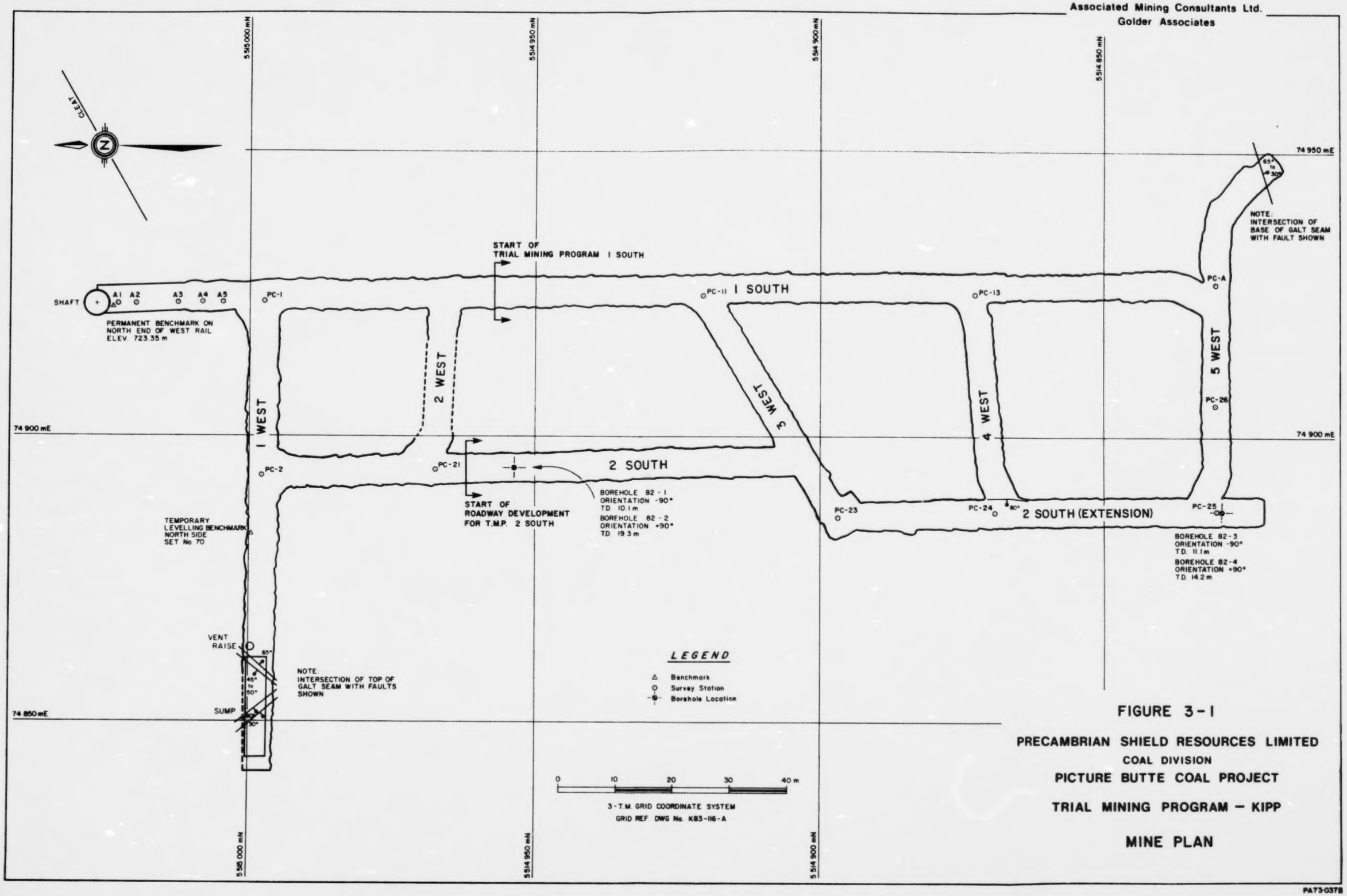

FIGURE 3-1

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

MINE PLAN

PA73-0378

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Golder Associates

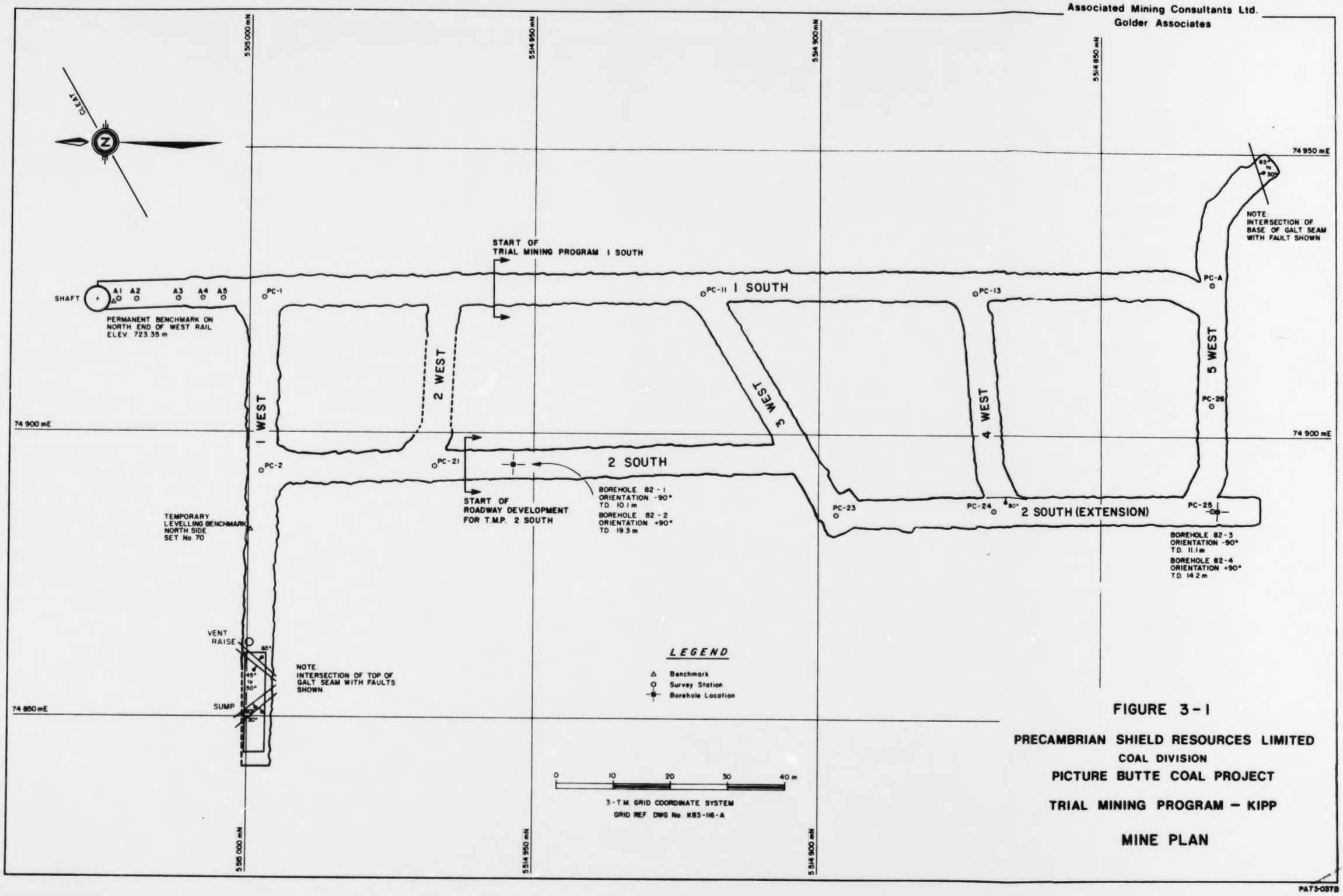


FIGURE 3-1

PRECAMBRIAN SHIELD RESOURCES LIMITED

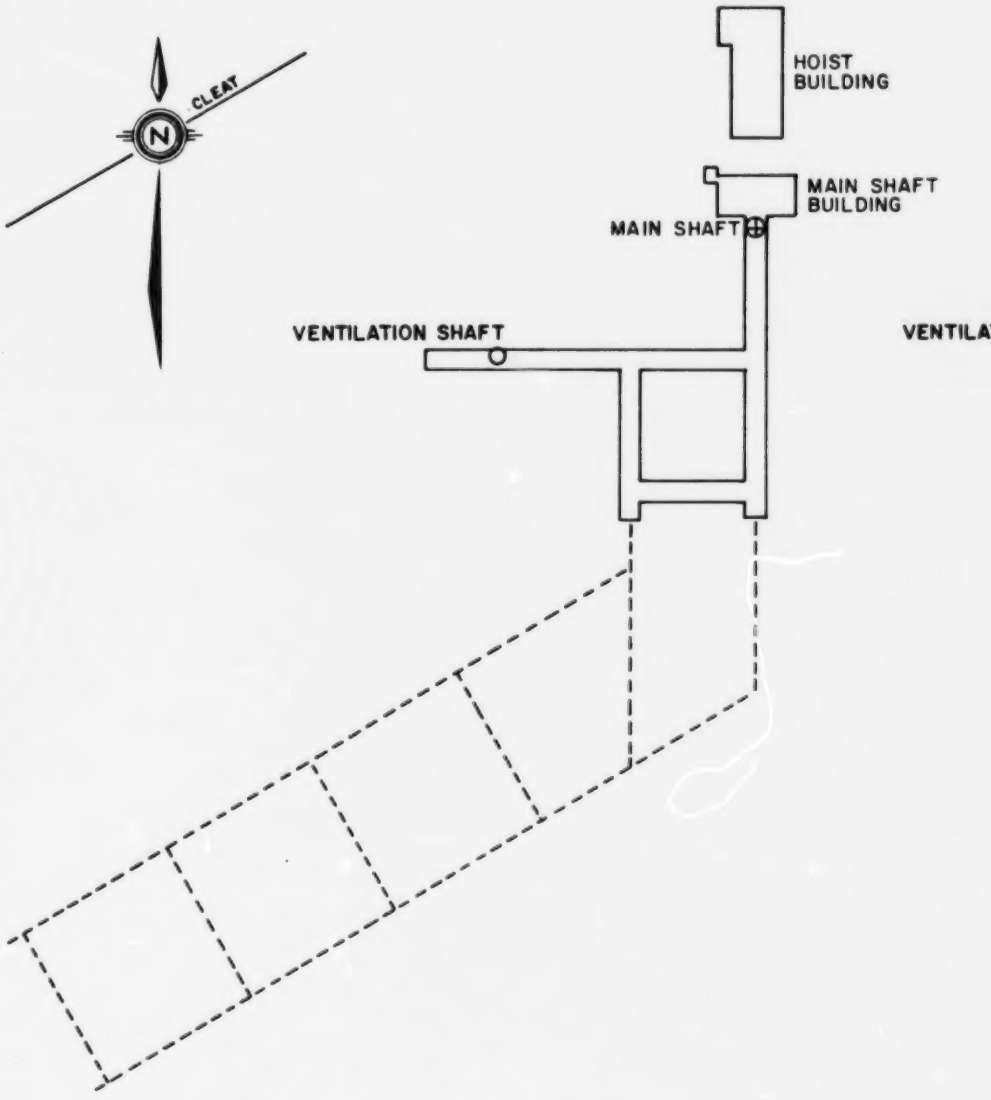
COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

MINE PLAN

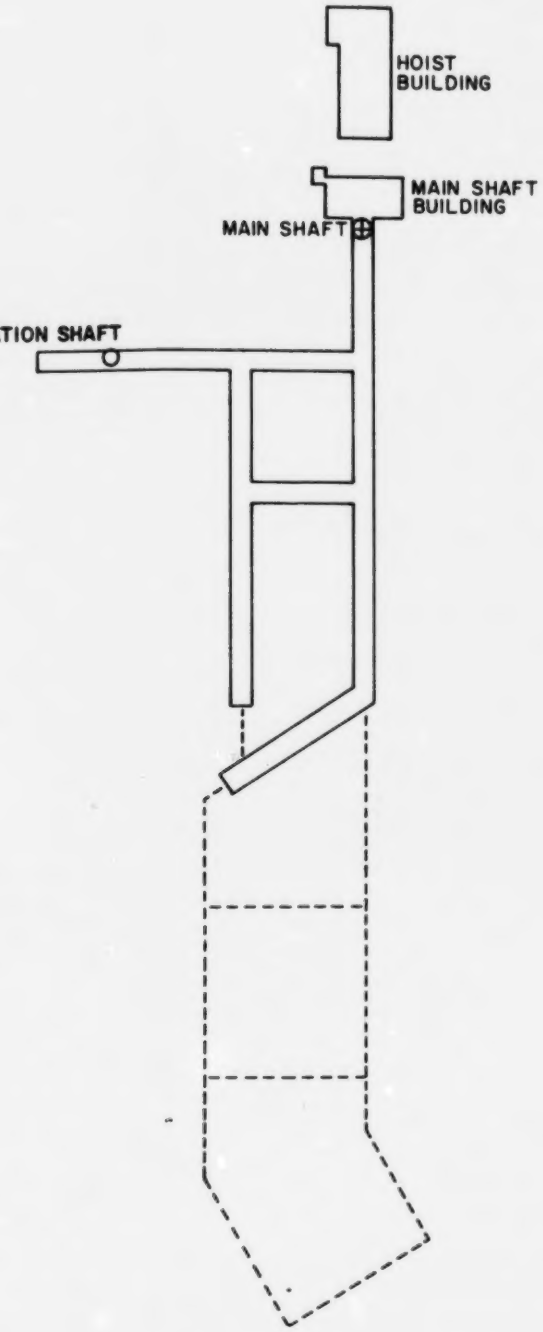
PA73-037B



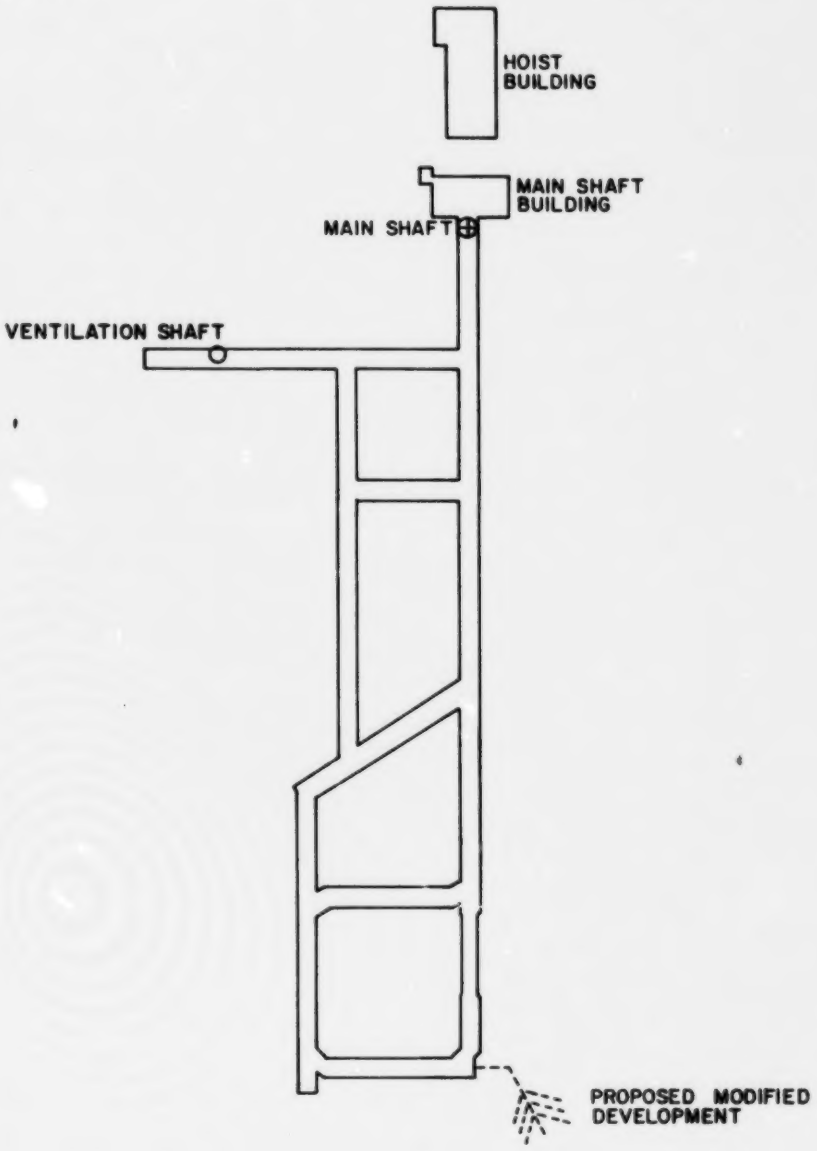
A. INITIAL PROPOSED MINE PLAN
MAY 6, 1982

EXISTING ROADWAYS

PROPOSED ROADWAYS



B. REVISED MINE PLAN
AUGUST 19, 1982



C. REVISED MINE PLAN
OCTOBER 8, 1982

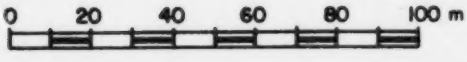
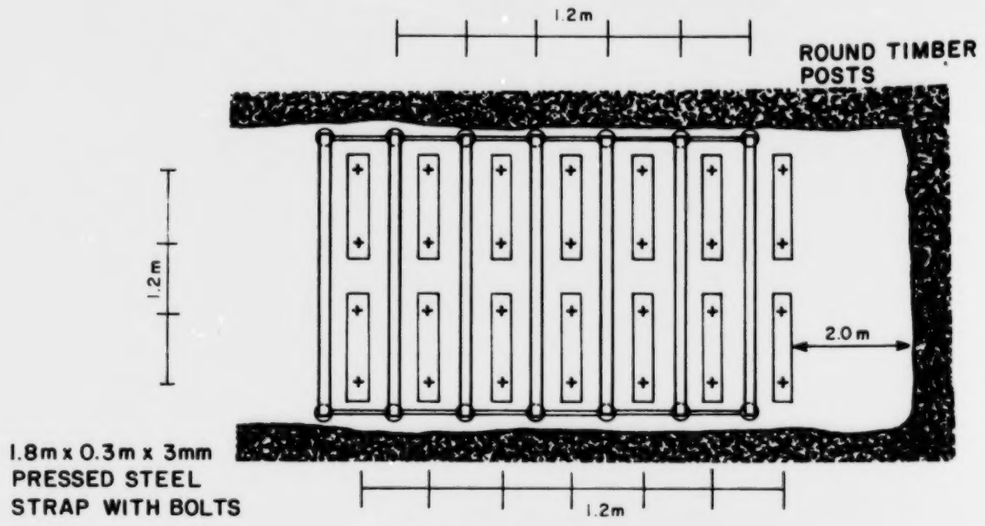


FIGURE 3-2

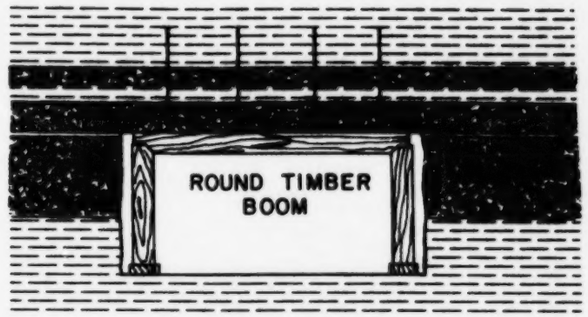
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

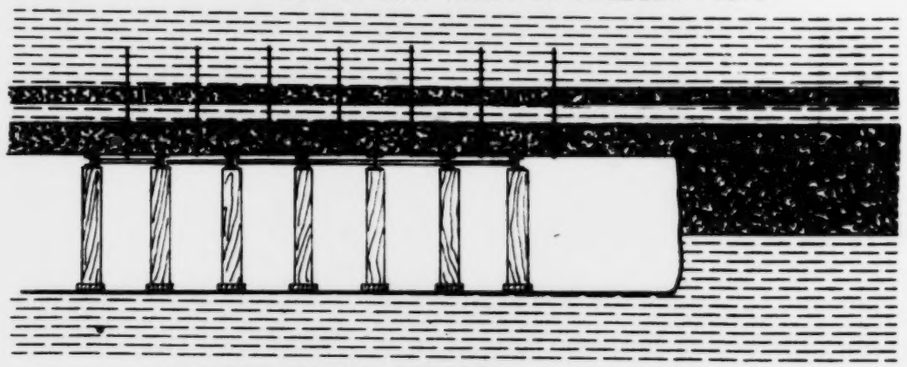
PROPOSED MINE PLANS



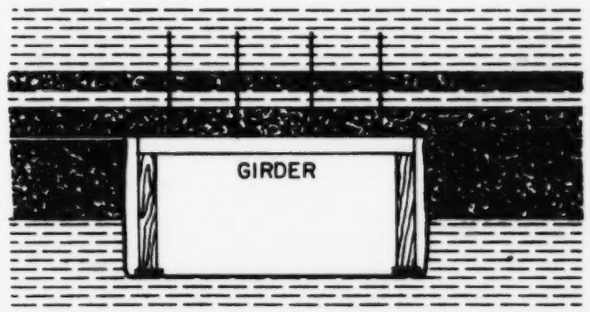
PLAN VIEW



CROSS SECTION



LONGITUDINAL SECTION



CROSS SECTION

N.T.S.

FIGURE 3-3

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

ROOF SUPPORT SCHEMATIC

PA73-007

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Golder Associates

74 950 mE

CLEAT

5 514 900 mN

5 514 850 mN

NOTE: DUE TO FAULT INTERSECTION ROCK BOLT PATTERN INTERRUPTED BETWEEN 5WEST SETS 51 & 56

1 SOUTH

1.8m REBAR BOLTS | 1.8m SWELLEX BOLTS | 1.8m REBAR BOLTS | 2.1m SWELLEX BOLTS | 2.1m REBAR BOLTS

START TMP 1 SOUTH

2 WEST

3 WEST

4 WEST

5 WEST

1.8m REBAR BOLTS

1.8m SWELLEX BOLTS

WIRE MESH PANELS

2.1m SWELLEX BOLTS

START TMP 2 SOUTH

1.8m REBAR BOLTS

2 SOUTH

2.1m REBAR BOLTS

74 900 mE

1.8m REBAR BOLTS | 2.1m REBAR BOLTS

2 SOUTH (EXTENSION)

5 514 950 mN

5 514 900 mN

LEGEND

- 8" x 6" x 35# (20 cm x 15 cm x 54 kg/m) RSJ's
- 6" x 5" x 25# (15 cm x 13 cm x 37 kg/m) RSJ's
- 20 cm x 20 cm Double H-Beam (PETRO-CANADA)
- Wood Boom
- BG Bearing Girder (Double H-Beam 12" x 8" x 70# 30 cm x 20 cm x 108 kg/m) RSJ's

0 5 10 15 20 25 m

3-TM GRID COORDINATE SYSTEM
GRID REF DWG No KB3-116-A

FIGURE 3-4

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM — KIPP

INITIAL SUPPORT PLAN

PA73-010A

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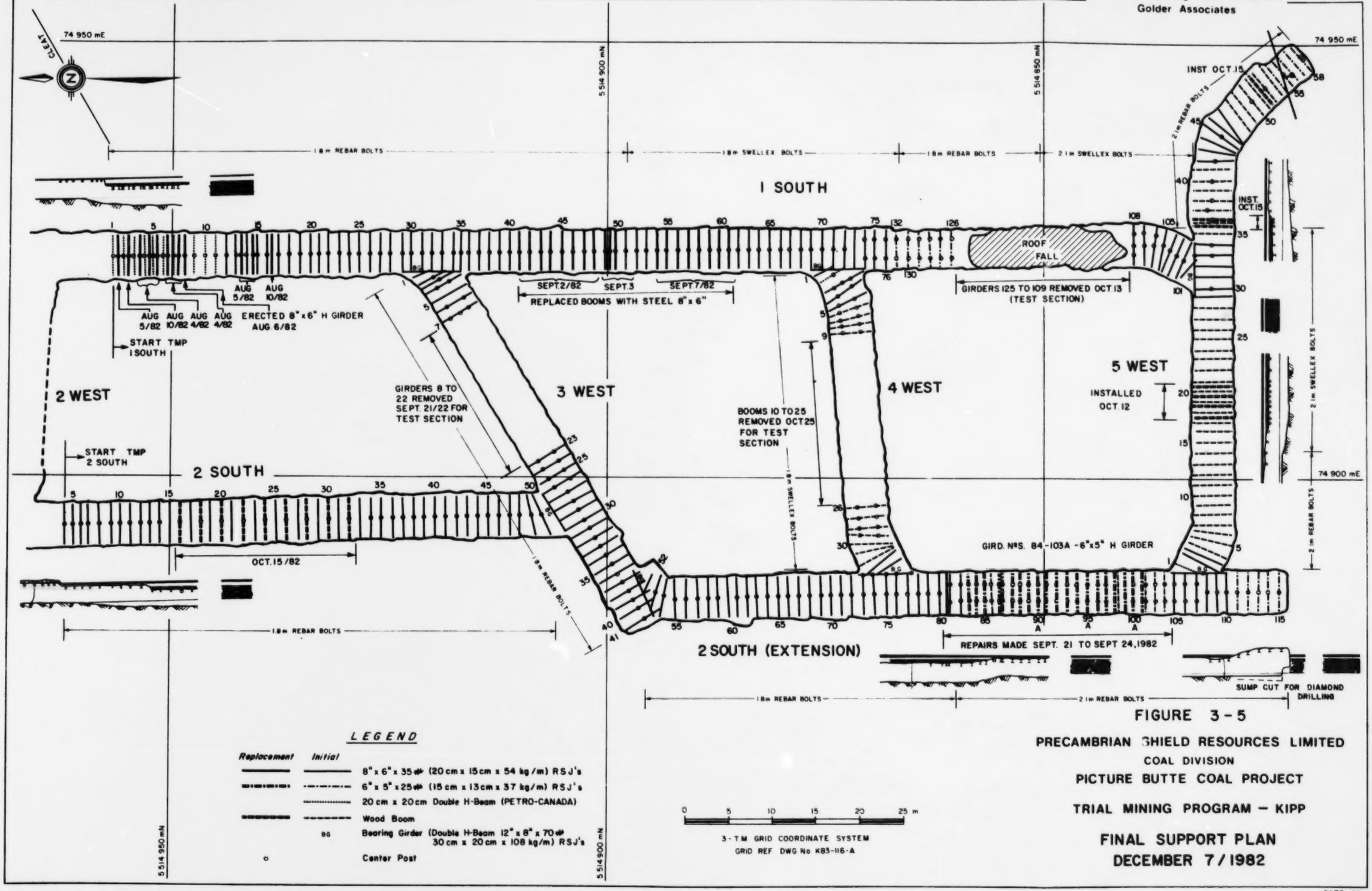
Golder Associates



LEGEND

Replacement	Initial	
———	———	8"x6"x35# (20cm x 15cm x 54 kg/m) RSJ's
———	———	6"x5"x25# (15 cm x 13cm x 37 kg/m) RSJ's
		20 cm x 20cm Double H-Beam (PETRO-CANADA)
———	———	Wood Boom
	BG	Bearing Girder (Double H-Beam 12"x8"x70# 30 cm x 20 cm x 108 kg/m) RSJ's
o		Center Post

0 5 10 15 20 25 m

3-TM GRID COORDINATE SYSTEM
GRID REF DWG No K83-116-A

FIGURE 3-5

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

FINAL SUPPORT PLAN
DECEMBER 7/1982

PA73-011A

Associated Mining Consultants Ltd.
Golder Associates

74 950 mE

74 900 mE

5514 950 mN

5514 900 mN

5514 850 mN

CLEAT

PREVIOUS MINING → TRIAL MINING PROGRAM (HORIZON CHANGE)

1 SOUTH

1 SOUTH

65° to 30°

HORIZON CHANGE

5.5 m LONG x 3m WIDE
INITIAL CUT LEFT UNSUPPORTED

5 WEST

2 WEST

3 WEST

CLEAT

4 WEST

HORIZON CHANGE

74 900 mE

HORIZON CHANGE

PREV. MINING → TRIAL MINING PROGRAM (HORIZON CHANGE)

2 SOUTH

2 SOUTH (EXTENSION)

HORIZON CHANGE

LEGEND

– Limit of Roof Spalling in excess of 15 cm, prior to installation of set

Overcutting into roof by the Continuous Miner is not shown

→ Direction of Mining

0 5 10 15 20 25 m

3-TM GRID COORDINATE SYSTEM
GRID REF DWG No K83-116-A

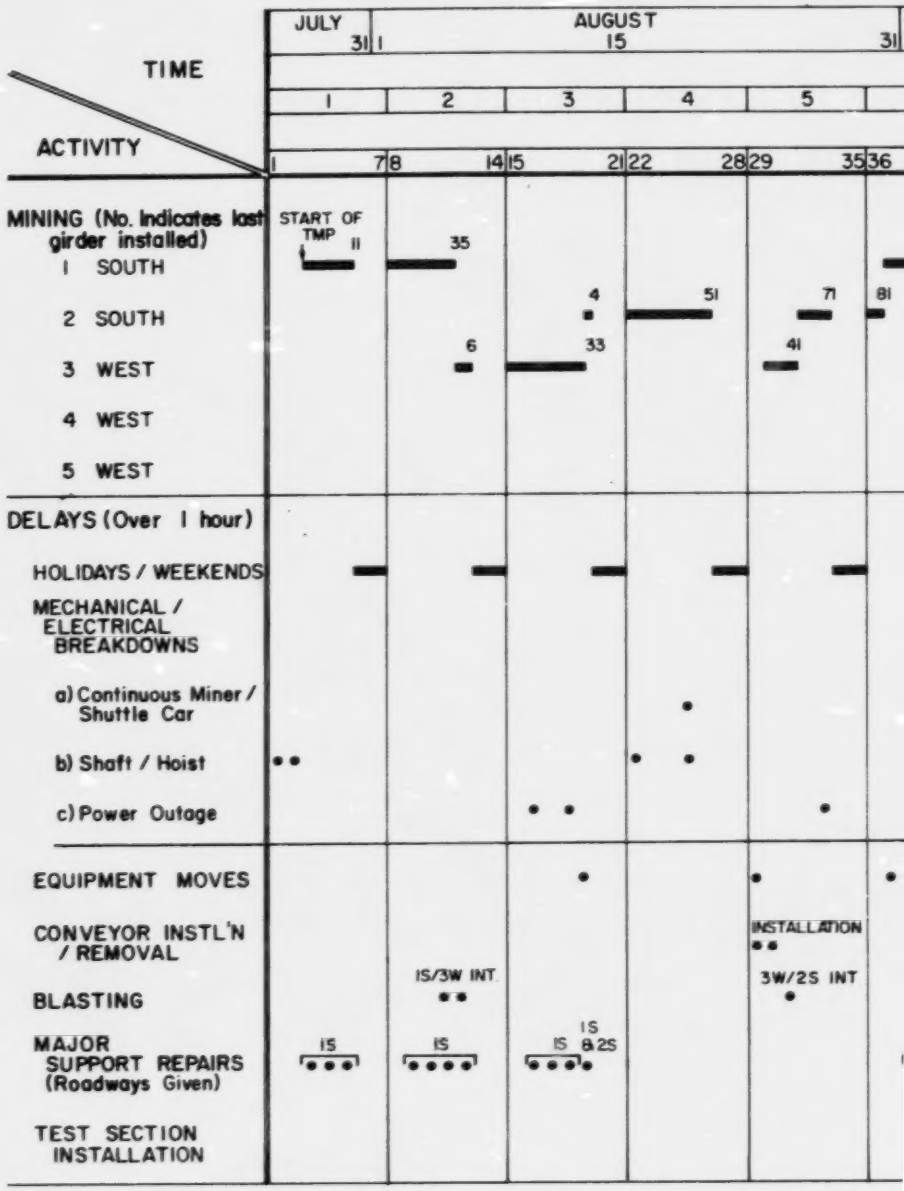
FIGURE 3-6

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

INITIAL ROOF STABILITY PLAN

PA73-013A


TIME
ACTIVITY
JULY
31
AUGUST
1
15
31
1
2
3
4
5
1
7
8
14
15
21
22
28
29
35
36
MINING (No. indicates last girder installed)
1 SOUTH
2 SOUTH
3 WEST
4 WEST
5 WEST
START OF TMP
11
35
4
51
71
81
6
33
41
DELAYS (Over 1 hour)
HOLIDAYS / WEEKENDS
MECHANICAL / ELECTRICAL BREAKDOWNS
a) Continuous Miner / Shuttle Car
b) Shaft / Hoist
c) Power Outage
EQUIPMENT MOVES
CONVEYOR INSTL'N / REMOVAL
INSTALLATION
BLASTING
1S/3W INT
3W/2S INT
MAJOR SUPPORT REPAIRS (Roadways Given)
1S
1S
1S
1S & 2S
TEST SECTION INSTALLATION

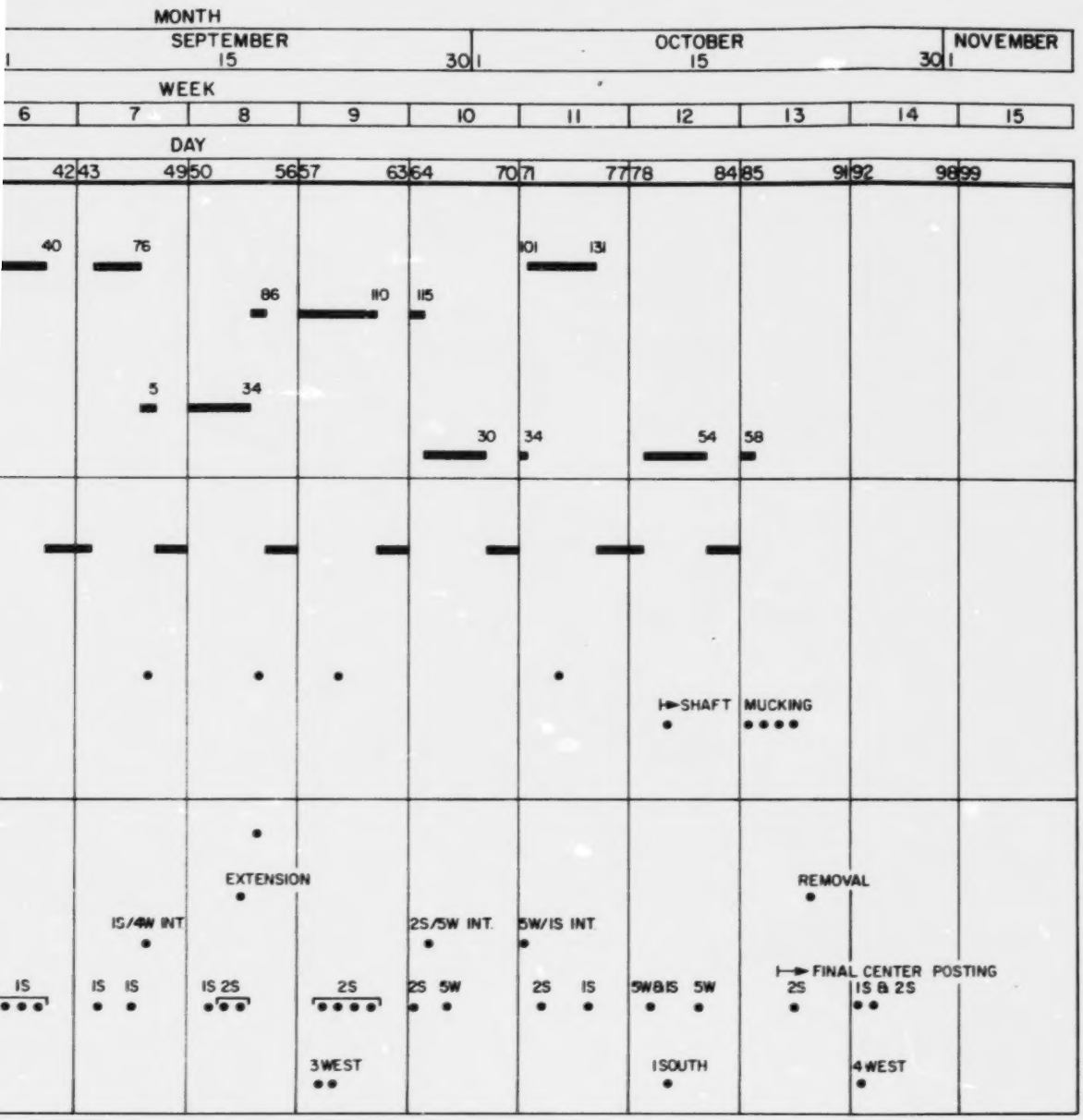


FIGURE 3-7

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

MINING ACTIVITY CHART

PA73-012

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SECTION 4 - INSTRUMENTATION

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.....Cont'd

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SECTION 4 - INSTRUMENTATION

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4-7 Girder and Extensometer Displacement - 2 South
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4-9 Instrumentation Graphs, Roof Extensometers 4W-20-1, 1S-53-1
4-10 Instrumentation Graphs, Floor Extensometers 4W-20-3, 1S-120-3
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4-12 Instrumentation Graphs, Load Cells 2S-19-2, 2S-60-2
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4-14 Final Instrumentation, November 23/1982
4-15 1 South Roof Fall

SECTION 4 - INSTRUMENTATION

4.1 INTRODUCTION

Physical measurements of strata behaviour during trial mining was designed to evaluate general deformation as well as specific support systems. Measurements chosen for the TMP included levelling of the roof, roadway convergence, roof-to-floor pressure, and roof and floor strata separation. Instrument design was simple and robust, while manual read-out ensured that instrument sites were visited frequently for visual inspections.

Measurement locations are given by roadway and set number (ie: 1 South - 72 refers to the area between Sets #72 and #73 in 1 South).

Instrumentation techniques were slightly different for installations during development as compared to those in test sections where some of the original support was withdrawn. Instrumentation of test sections was phased so as not to interfere with mining operations.

4.2 INSTRUMENTATION

4.2.1 Survey Levels

Levels were run between August 31 and November 24, 1982. The purpose of these levels was to establish overall roof subsidence, deflection on selected H-beams, and movement on roof extensometers. A temporary benchmark was established in 1 West and used throughout the TMP (Figure 3-1). Overall, the accuracy of the levelling is in the order of 1 cm for intermediate points.

4.2.2 Extensometers

Single rod extensometers were installed in the roof routinely during development. These were 2.4 m Rebar bolts installed in a 2.2 m stoper hole with one 0.3 m resin cartridge. These installations were made immediately beside a roof bolt plate near the centre of the roadway. Bed separation within 2 m above the roof was measured with a measuring tape between the roof bolt head and adjacent plate to an accuracy of ± 2 mm.

Installation in the test sections varied slightly. In addition to a roof bolt, PVC tubing was cemented into the collar of an up or down hole and a target calibrated with 1 cm markings was attached to this collar. The relative position between roof bolt head and target could then be observed from a safe location to an accuracy of ± 5 mm.

Forty-nine roof extensometers and 9 floor extensometers were installed during the TMP.

4.2.3 Load Cells

Roof-to-floor pressure measurements are early indicators of roof or floor deformation. Hydraulic rams (load cells) with 152 mm diameter pistons and a very short stroke were mounted on wooden posts either against the roof or under the ends of steel girders. A short length of high pressure hydraulic tubing was attached to each load cell, fitted with a 20.7 MPa (3000 psi) pressure gauge, a high pressure valve and a quick connect coupling. The load cells were set on their posts, shimmed with wood against the roof or H-beam and preloaded to 3.5 MPa (500 psi). As a safety precaution each load cell was chained to the closest roof bolt plate.

Twelve load cells were installed in 17 locations during the TMP.

4.2.4 Convergence Rods

The convergence rods used to measure vertical roadway deformation were a very simple design. Two telescoping sections of electrical conduit were held apart by an internal compression spring. Convergence reading could be taken directly with a measuring tape by measuring the distance between a notch in the smaller conduit and the end of the larger conduit to an accuracy of ± 2 mm.

Installation in the test sections included a target calibrated with 1 cm markings attached to the smaller conduit to an accuracy of ± 5 mm.

Twenty convergence rods were installed in 30 locations during the TMP.

4.2.5 Rib-to-Rib Convergence

Rib-to-rib convergence stations were set up by cementing a 30 cm wooden plug into short boreholes on opposite sides of the roadway. Hooks were installed in the plugs and the distance between them measured with a tape to an accuracy of ± 10 mm.

Nine rib-to-rib convergence stations were set up during the TMP.

4.3 INSTRUMENT MONITORING AND RESULTS

The instrumentation described above was installed throughout the TMP. Figure 4-1 shows all the instrumentation installed under full roof support. Three test sections were established where the sets were withdrawn, leaving only the roof bolt support. These were instrumented

extensively to monitor strata reaction and were located in 3 West (Figure 4-2), 1 South (Figure 4-3) and 4 West (Figure 4-4).

4.3.1 Survey Levels

Levelling was carried out from late August to November 24, 1982. Most data was obtained from 2 South, 4 West and 5 West. Although some information was obtained from 1 South, this roadway was centre-posted early in the TMP, making the results less meaningful.

Generally, there was very little roof sag throughout the levelled headings. Figure 4-5 illustrates the minimal deflection of the roof in 4 West over a 69 day period, including the last 29 days with Swellex roof bolt support only. Most of the elevation changes throughout the levelled area of the mine fall within ± 2 cm of the base line elevation.

2 South roof deflection between Set #57 and the junction with 4 West exhibits a classic case of roof sag with central tension cracks. The roof sag increases from 2 cm at Set #55 to 12 cm at Set #72 in a gradual manner. Figure 4-6 illustrates this change with three longitudinal-sections, one along each rib and one along the centre-line. The roof in 2 South sheared off across this junction with 4 West (Set #73 to Set #77)along the east rib. However, the roof sag along the centre-line Set #73 to Set #82 follows the trend established further north between Set #55 to Set #72 (Figure 4-6) .

This same relationship is also illustrated in Figure 4-7 with 5 sections along selected girders. The increasing central sag is quite evident from Set #14 to Set #65, Set #77 shows the

sheared-off east rib and Set #105 shows a return to a slight deflection along this roadway.

1 South near the junction with 5 West is one other area where roof sag was documented by levelling. Set #103 to Set #105, monitored over 28 days, showed a maximum subsidence of 2 cm along the west rib, 3 cm along the east rib and 10 cm along the centreline.

Levels on roof extensometer bolt heads were obtained for almost all extensometers. The results of these levels are discussed under that heading.

4.3.2 Extensometers

Roof extensometers were installed under all types of roof support. Survey levels were run separately on the extensometer heads. The period of observation for the 49 extensometers varied from 105 days to 25 days, resulting in hundreds of observations.

Roof extensometers installed during TMP give an excellent record of bed separation. Where installed in triplets anchored at 1.2 m, 2.4 m and 3.6 m, the total bed separation can be assigned to separate strata intervals in the roof (ie: bed separation from 1S-120-1A - 11 cm, 1B - 7 cm and 1C - 3 cm, Figure 4-8):

Bed Separation	Roof Strata Interval
3 cm	Mine roof to 1.2 m
4 cm	1.2 m to 2.4 m
4 cm	2.4 m to 3.6 m

Any drop in elevation of the extensometer head would be interpreted as bed separation above the extensometer anchor.

Each individual time vs deformation graph has its own particular pattern. However, these patterns can be grouped based on the initial rate of bed separation and the total bed separation. Figures 4-8 and 4-9 illustrate the extensometer graphs.

a. Initial Bed Separation.

The slope of each time vs deformation graph was determined between the time of installation and the first major inflection point. All graphs that exhibited a distinct initial slope also had a well defined inflection point. The initial rates of bed separation were grouped into three ranges:

low	less than 0.1 cm/day
intermediate	0.1 cm to 0.5 cm/day
high	greater than 0.5 cm/day

Table 4-1 lists the extensometers and their grouping.

The low rates of bed separation were found in all of 4 West, almost all of 3 West and in the northern part of 2 South to Set #30. In addition, 1S-31-1, in the 1 South/3 West junction, and 1S-111-1C, the 1.2 m extensometer in 1 South Test Section, are also included.

The intermediate group includes one third of the extensometers in 1 South, half of those in 2 South including 2S-77-1 in the 4 West junction, two in 3 West and two in 5 West.

The high rates of bed separation were found in over one half of the extensometers in 1 South, two in 2 South where no roof coal was left, in 4 West the closest extensometer to the 2 South

junction, and three in 5 West, including two where no roof coal had been left.

Generally this grouping reflects the roof conditions encountered soon after mining.

b. Total Bed Separation

The total bed separation at any site comprises the bed separation below the extensometer anchor, measured by the extensometer, and bed separation above the anchor determined by the survey levels (ie: change in elevation of the extensometer head). Where no levels are available the total bed separation shown in Table 4-1 is a minimum value. Again, the bed separations were split into three levels:

low	less than 2.5 cm
intermediate	2.4 cm to 5.0 cm
high	greater than 5.0 cm.

The low total bed separations were found in all of 3 West, 4 West, 2 South as far south as Set #52, 5 W-22-1 in 5 West and 1S-111-1C (a 1.2 m extensometer) in Test Section 1 South.

The intermediate group includes three in 1 South, two in 2 South where no roof coal had been left, four in 5 West and in 4 West the closest extensometer to the 2 South junction.

The high total bed separations were found in almost all extensometer locations in 1 South, and two in 2 South either in or close to the 4 West junction.

c. Discussion

The time over which extensometers were monitored varied between 25 and 105 days. It was, therefore, not expected to find a correlation between the initial rate of bed separation and the total bed separation for these diverse time intervals. Nevertheless there appears to be an excellent correlation of these two parameters (Table 4-1) in that a low initial rate of bed separation indicates a low total bed separation (in the readings proportion 20:19) while a high initial rate indicates a high total bed separation (in the readings proportion 15:9). Overall 65% of the extensometers give a direct correlation while the 17 that do not conform were distributed as follows:

- Six initial rates predicted less than the achieved total bed separation (one in 1 South/3 West junction, two in the 2 South/4 West junction and three in or near the Test Section 1 South.
- Eleven initial rates predicted greater total bed separation than achieved (six of these are associated with the higher mining horizon in 2 South and 5 West).

The roof extensometers have provided an excellent record of bed separation. Continued use of this type of instrumentation under similar conditions should provide a good predictability of roof stability.

Floor extensometers have shown very little bed separation. Of the 9 installations only the two in 1 South Test Section showed any bed separation whatsoever (Figure 4-10). This may well be a function of timing. Floor extensometers were installed much later than roof extensometers (ie: 41 days later in 4 West). Test Section 1 South was installed 5 days after mining. Floor heave, therefore, appears to take place soon after mining.

4.4.3 Convergence

Convergence was measured and indicated by various instruments:

- hydraulic load cells
- telescoping convergence rods
- rib-to-rib measurements

a. Load Cells

Load cells are very good indicators of small deformations in either roof or floor. Each load vs time graph indicates a cyclic loading and unloading curve.

We attempted to correlate these load changes to extensometer readings or distance to mining but neither shows any correlation. It appears that some of the cycles seem to be synchronous. This would require a broader influence than local strata deformation.

The level of loading, however, differs markedly. Load cell readings in 3 West and 4 West had a range up to 4.8 MPa and in 2 South up to 11.0 MPa with the western cell always being the higher. In 1 South the eastern cell was always the higher, up to 14.5 MPa and this load was only exceeded in Test Section 1 South where the maximum was 18.6 MPa. These very high loads definitely exceeded the bearing strength of the roof coal evidenced by the 20 cm diameter load cell pushed into the roof. Figures 4-11 and 4-12 illustrate load cell instrumentation in the test areas in 1 South and 2 South.

The load cells gave excellent indication of early strata pressures, but the variations in their loads could not be related to any activity in the mine.

b. Convergence Rods

Convergence rods in Test Sections 3 West and 4 West show virtually no movement after supports had been removed.

In 2 South 2S-70-4 near the 4 West junction shows an excellent deformation curve of decellerating closure in 2 South (Figure 4-13). One convergence rod in Test Section 1 South (1S-120-4) located east of the centre-line (Figure 4-3) shows an ever decreasing deformation rate until six days before the roof fall, at that time the curve steepens drastically (Figure 4-11). Another convergence rod (1S-123-4) less than 3 m away but on the west of the centre-line (Figure 4-3) does not show this increased rate of deformation.

c. Rib-to-Rib

Horizontal closure has not been demonstrated during the TMP. If rib-to-rib closure has taken place, it is very limited and would not affect mining.

4.4.4 Final Instrumentation

The installed instruments were monitored by PSRL technicians on a shift by shift basis until October 27 and on a daily basis until November 19. Data acquisition ceased as of November 23 for the purposes of this report. Final location of instrumentation is shown in Figure 4-14.

4.4.5 Roof Fall

Beginning on November 22, 1982 a series of roof falls occurred in 1 South. Extent of the fall area is illustrated in Figure 4-15. We

are aware of three different roof falls, the first of which occurred on November 22, 1982. This minor fall was estimated to be only about 0.3 m high and covered an area of 1.2 m x 2.4 m. It was noted immediately after the fall that several roof bolts (Swellex type) had broken. The east portion of the straps were held by secured bolts located against the east rib.

The major roof fall occurred between day shifts on November 22 and 23, 1982. It was noted that some roof support straps at both ends of the fall area were were still in place. At both ends of the fall in I South some roof bolts (both Rebar and Swellex) along the rib line were still in place (Figure 4-15). Several broken Swellex bolts were recovered at the south end of the fall area. The height of the fall could only be estimated, due to safety considerations, however judging from the intersected roof strata that was exposed, the maximum height of the cave was about 5 m above the existing roof.

After this second fall it was noted that the post supporting load cell IS-109-2 was kicked out towards the south of the roadway. This observation indicates that the fall likely started in the centre of the fall area and extended towards the end.

The third observed roof fall occurred on December 7, 1982, where the unsupported lip immediately north of Set #108 fell. The vertical extent of this fall could only be estimated due to limited access for safety considerations.

Monitoring of roof conditions in the I South area prior to installation of the test section was done with two load cells (IS-117-2A and B) and two roof extensometers (IS-114-1 and IS-128-1). These instruments indicated a lower rate of roof loading and bed

separation than found when driving south in I South. This lower rate was probably due to the narrowing of I South roadway.

On October 13, instrumentation was installed in the Test Section and roof support was withdrawn. Instrumentation readings indicated, almost immediately, substantial convergence and bed separation in the roof between 2 m and 3.2 m above the top of the Galt Seam (Figure 4-10). After October 18, the rate of deformation reduced slightly. However, between November 10 and 15, the rate of deformation indicated by a number of instruments increased.

Roof bolts alone are not capable of supporting the roof in north-south roadways. Deformation and bed separation in north-south headings are arrested by steel sets. If sets are removed in these headings the rate of deformation and bed separation will increase to the point of roof failure, as it did in Test Section I South.

Another factor which could have contributed to the Test Section I South failure was the presence of water, observed only as dampness on the east rib of I South. The extent of the water, and therefore the effects of the water on Test Section I South are not known.

TABLE 4-1
PICTURE BUTTE COAL PROJECT

BED SEPARATION MEASURED BY ROOF EXTENSOMETERS AND SURVEY LEVELS

ROOF EXTENSOMETER	INITIAL BED SEPARATION[4] RATE (cm/day)			TOTAL BED SEPARATION[#] EXTENSOMETER AND ΔELEVATION (cm)		
	<0.1	0.1-0.5	>0.5	<2.5	2.5-5.0	>5.0
1S- 31-1	0.02				4.1	
1S- 40-1			0.55			6.7†
1S- 53-1			0.90			7.0†
1S- 58-1			0.88			6.7†
1S- 63-1		0.13			4.0	
1S- 67-1A			0.76			11.2
1S- 67-1B			1.00			10.0
1S- 67-1C			0.72			9.8
1S-128-1		0.48				6.0•
1S-120-1A			1.30			11.0†•
1S-120-1B			1.10			7.0•
1S-120-1C		0.29			3.0•	
1S-114-1		0.43				24.3•
1S-111-1A		0.43				11.5•
1S-111-1B			0.80			11.4†•
1S-111-1C	0.04			1.6•		
2S- 19-1	0.00			0.2		
2S- 24-1	0.01			0.5		
2S- 29-1	0.00			0.3		
2S- 46-1		0.14		1.4		
2S- 52-1		0.33		1.3		
2S- 56-1		0.36			3.6†	
2S- 68-1		0.20				9.3†
2S- 77-1		0.37				17.5†
2S- 87-1			0.83		3.8•	
2S- 94-1			1.07		3.2•	
3W- 4-1		0.50		1.5		
3W- 11-1A	0.00			0.0		
3W- 11-1B	0.00			0.0		
3W- 11-1C	0.00			0.0		
3W- 18-1A	0.00			0.0		
3W- 18-1B		0.12		1.7		
3W- 18-1C	0.00			0.0		
3W- 29-1	0.00			1.5		
4W- 2-1	0.00			1.7		
4W- 8-1	0.00			0.5		
4W- 14-1A	0.00			0.0		
4W- 14-1B	0.00			0.0		
4W- 14-1C	0.00			0.0		
4W- 19-1	0.01			0.5		
4W- 20-1A	0.00			0.0		
4W- 20-1B	0.00			0.0		
4W- 20-1C	0.00			0.0		
4W- 24-1			1.25		4.2	
5W- 9-1			0.80		4.8•	
5W- 13-1			0.75		4.5	
5W- 19-1			0.80		4.7	
5W- 22-1		0.28		1.5		
5W- 32-1		0.36			4.0	

[4] Initial bed separation usually measured over 3 to 8 days.
[#] Total bed separation includes extensometer reading plus the drop in elevation.
† Some Extensometer bolts became tight in their boreholes because of strata deformation. Total bed separation for these bolts without levels would be a minimum value.
• No levels were run. Total bed separation is a minimum estimate.

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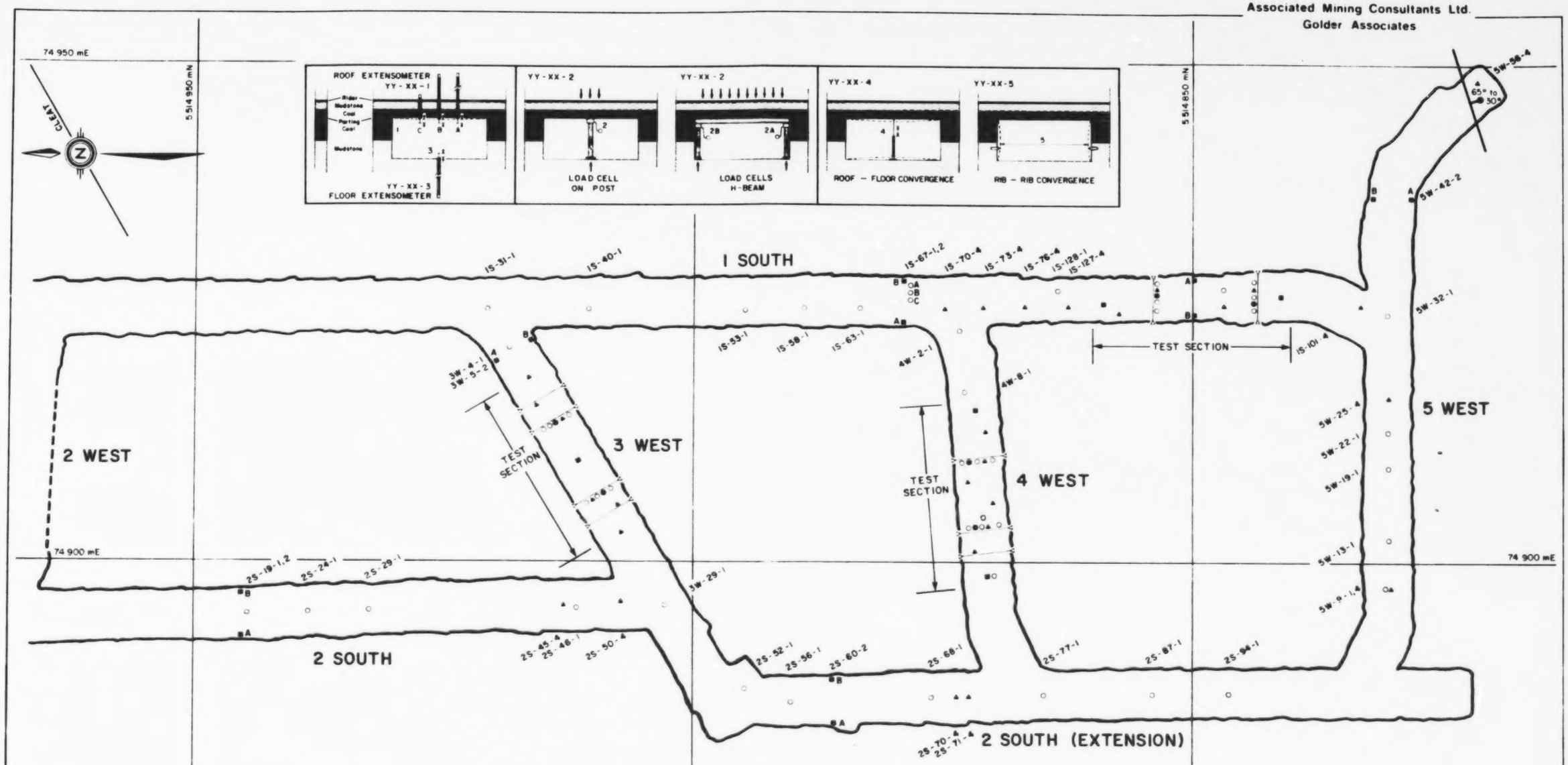


INSTRUMENTATION

SYMBOL	NAME	CODE
○	ROOF EXTENSOMETER	YY - XX - 1
■	LOAD CELL	YY - XX - 2
●	FLOOR EXTENSOMETER	YY - XX - 3
▲	CONVERGENCE ROD	YY - XX - 4
⊢⊣	RIB - RIB CONVERGENCE	YY - XX - 5

YY - DENOTES ROADWAY No
XX - DENOTES GIRDER No

0 5 10 15 20 25 m

3 TM GRID COORDINATE SYSTEM
GRID REF DWG No KB3-116-A

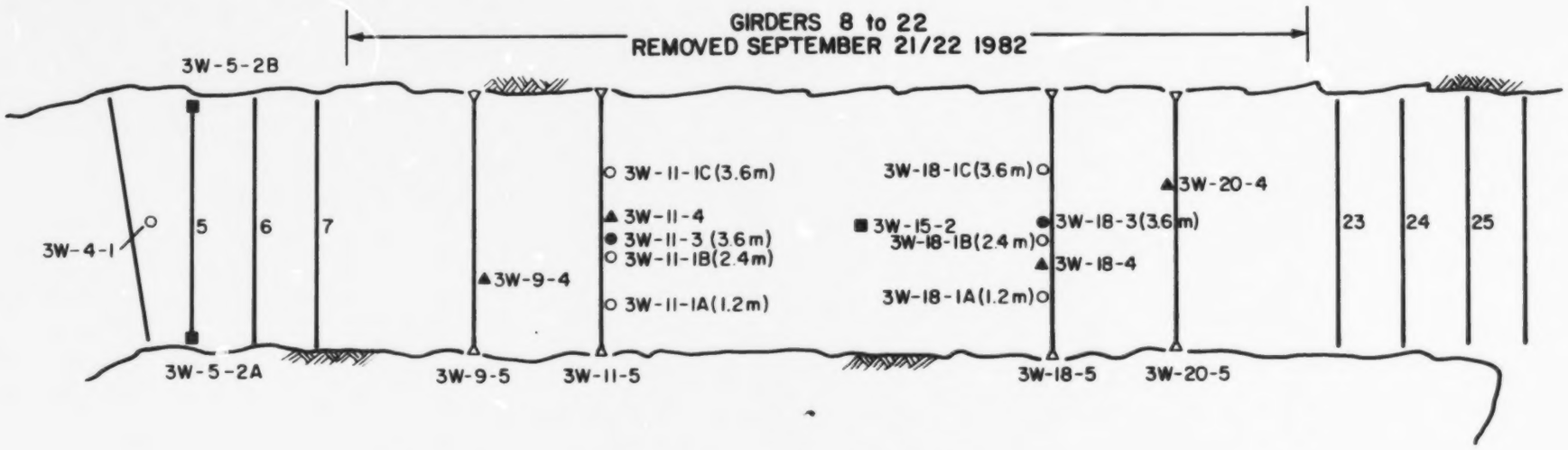
FIGURE 4 - 1

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

TOTAL INSTRUMENTATION

PA73-068A



3WEST ROADWAY

NOTE: ALL GIRDERS 8"x 6"x 35#x 4.6m
ALL BOLTS 1.8m REBAR ON 1.2m PATTERN

INSTRUMENTATION

SYMBOL	NAME	CODE
o	ROOF EXTENSOMETER	3W-XX-1
■	LOAD CELL	3W-XX-2
●	FLOOR EXTENSOMETER	3W-XX-3
▲	CONVERGENCE ROD	3W-XX-4
▷—◁	RIB-RIB CONVERGENCE	3W-XX-5

XX - DENOTES GIRDER No.

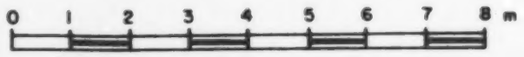


FIGURE 4-2

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

DETAILED TEST SECTION
3 WEST

PA73-018

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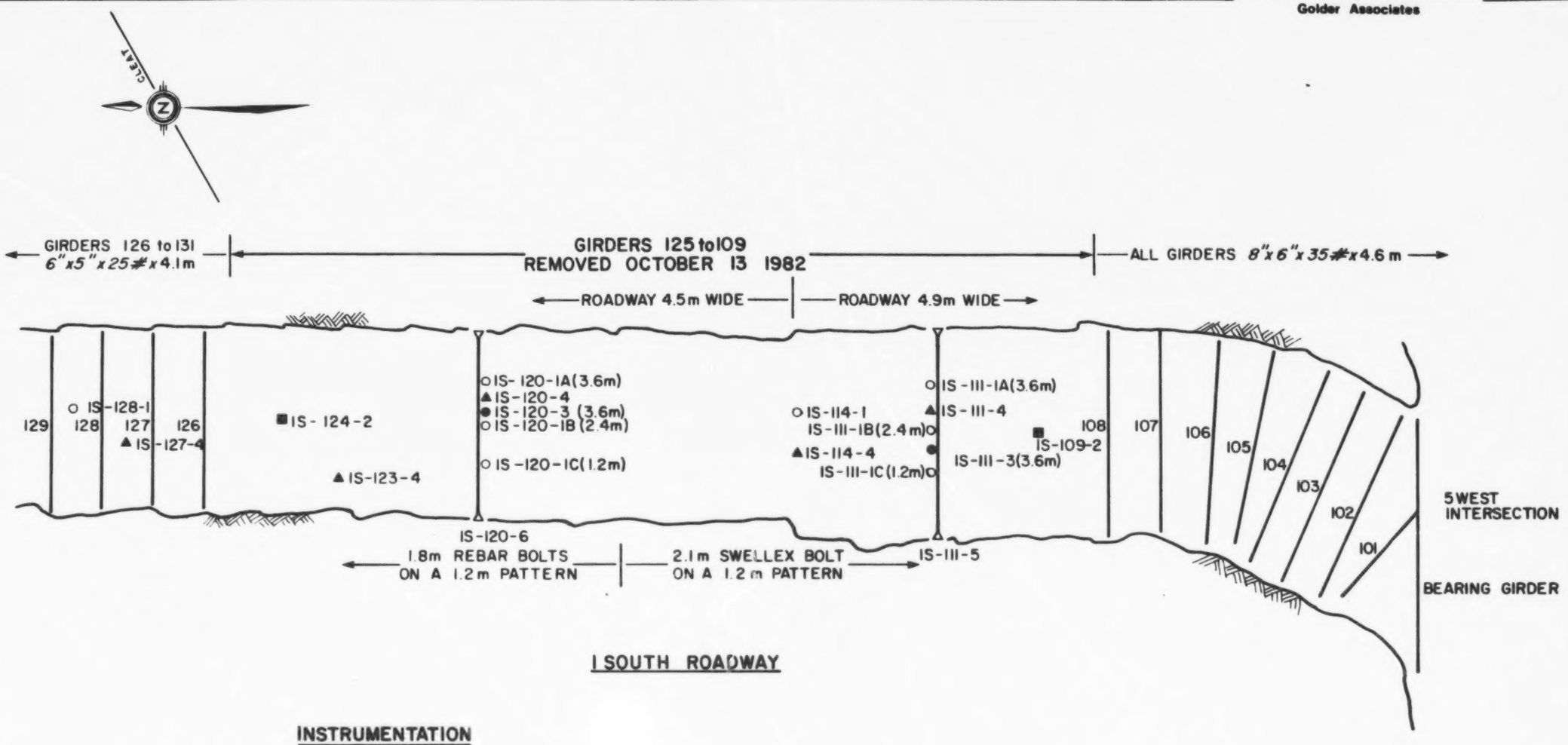

I SOUTH ROADWAY

INSTRUMENTATION

SYMBOL	NAME	CODE
○	ROOF EXTENSOMETER	IS-XX-1
■	LOAD CELL	IS-XX-2
●	FLOOR EXTENSOMETER	IS-XX-3
▲	CONVERGENCE ROD	IS-XX-4
▷—◁	RIB-RIB CONVERGENCE	IS-XX-5

XX-DENOTES GIRDER No.

0 1 2 3 4 5 6 7 8 m

FIGURE 4-3

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM — KIPP

DETAILED TEST SECTION
I SOUTH

PATS-004

Associated Mining Consultants Ltd.
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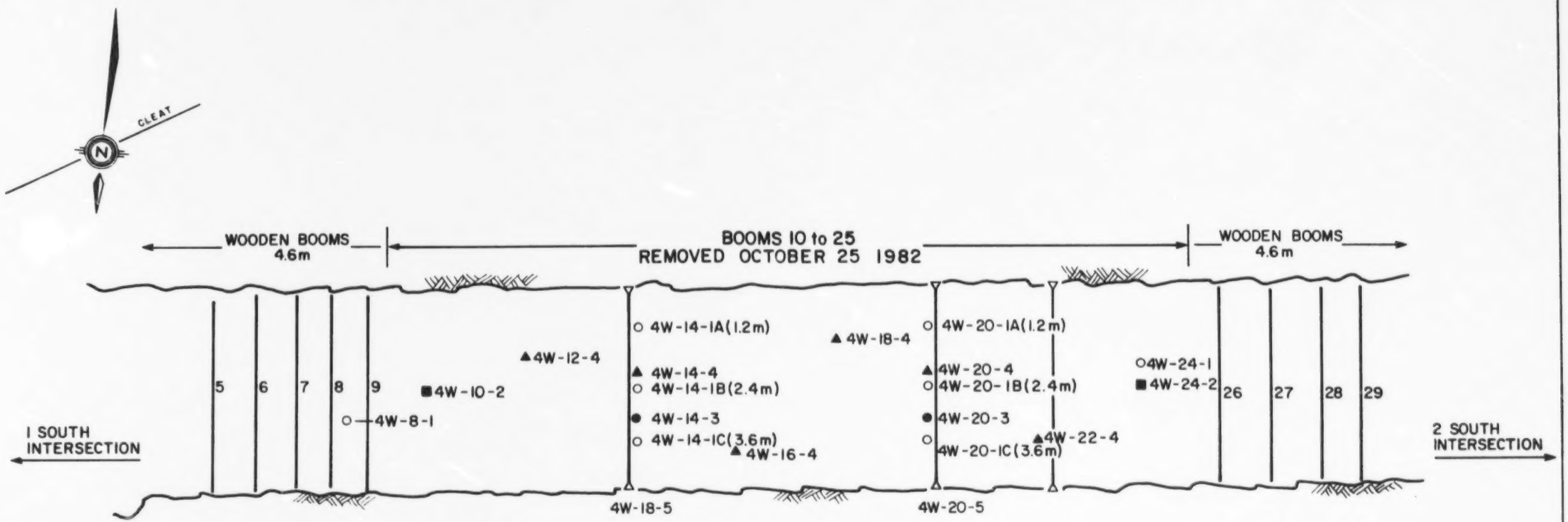

4 WEST ROADWAY

INSTRUMENTATION

SYMBOL	NAME	CODE
○	ROOF EXTENSOMETER	4W-XX-1
■	LOAD CELL	4W-XX-2
●	FLOOR EXTENSOMETER	4W-XX-3
▲	CONVERGENCE ROD	4W-XX-4
▷——◁	RIB-RIB CONVERGENCE	4W-XX-5

XX - DENOTES GIRDER No.

NOTE: ALL BOLTS 1.8m SWELLEX ON 1.2m PATTERN

0 1 2 3 4 5 6 7 8 m

FIGURE 4-4

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP

DETAILED TEST SECTION
4 WEST

PA73-016

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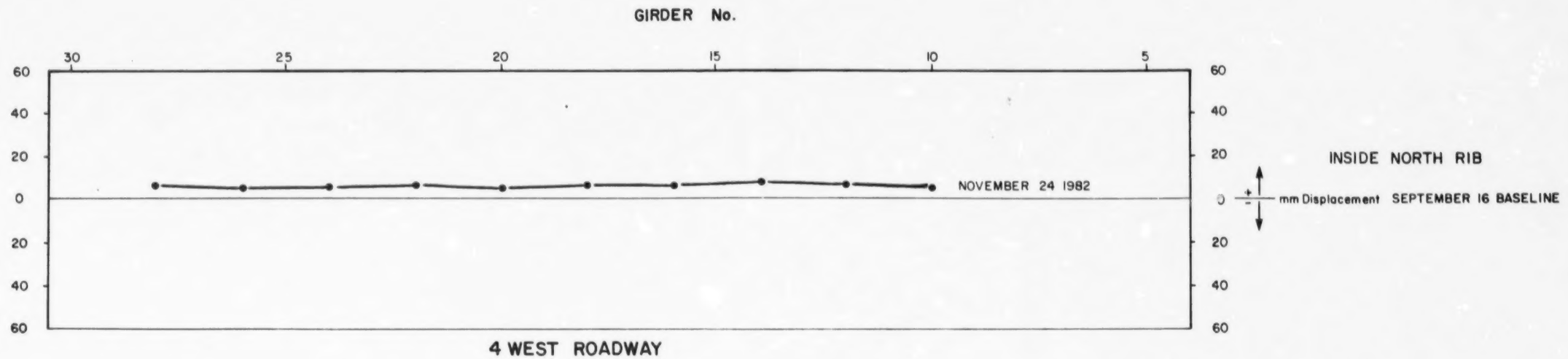


NOTE: LEVELS TAKEN ON SWELLEX BOLT HEADS,
TO THE NORTH OF CENTRE LINE.

FIGURE 4-5

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

ROOF SUBSIDENCE LEVELS
4 WEST

PA73-041

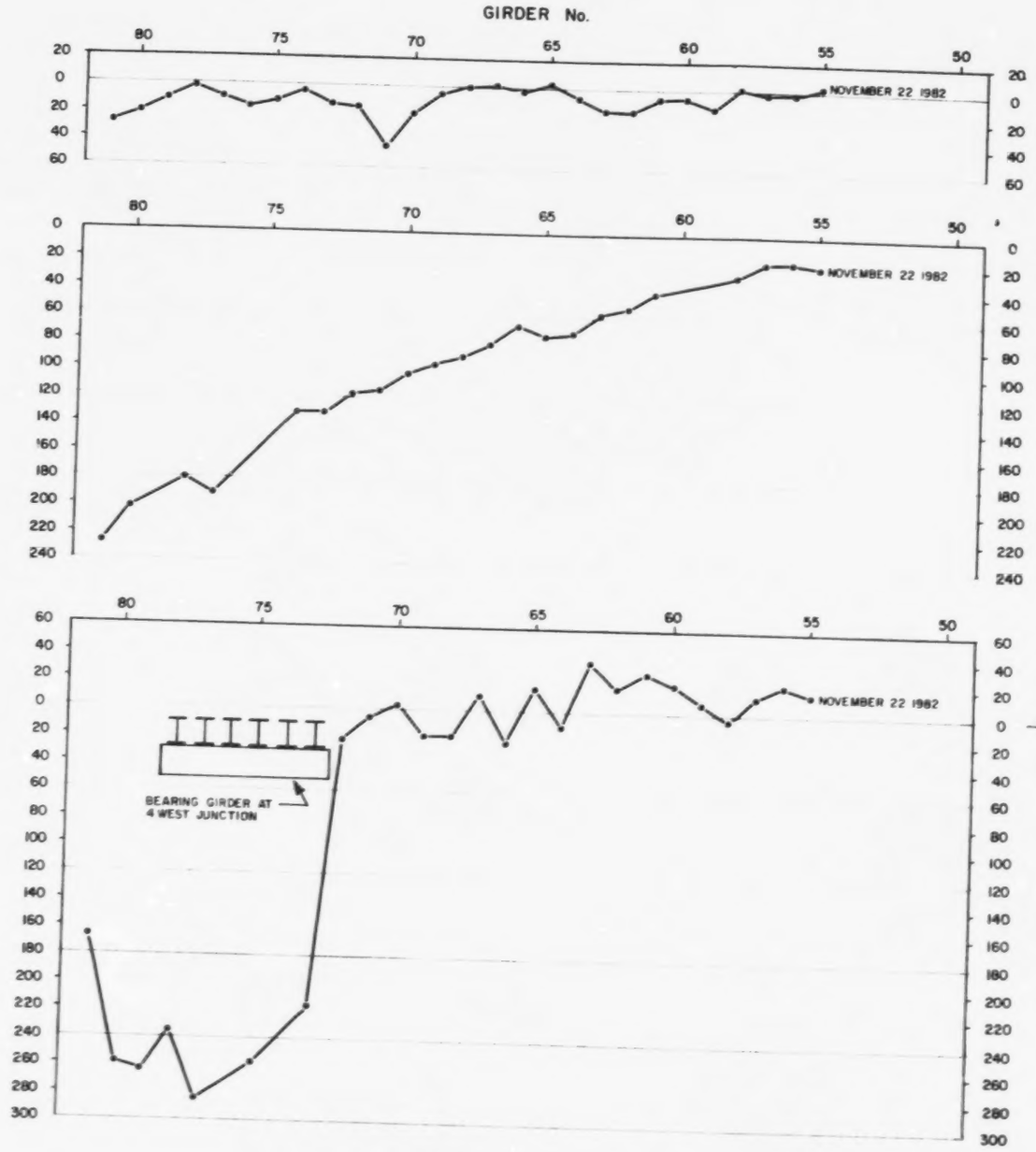

GIRDER No.
80 75 70 65 60 55 50
NOVEMBER 22 1982
+ - mm Displacement AUGUST 31 BASELINE
GIRDERS ALONG WEST RIB
NOVEMBER 22 1982
- mm Displacement AUGUST 31 BASELINE
GIRDERS ALONG CENTRE LINE
NOVEMBER 22 1982
BEARING GIRDER AT 4 WEST JUNCTION
+ - mm Displacement AUGUST 31 BASELINE
GIRDERS ALONG EAST RIB
2 SOUTH ROADWAY

FIGURE 4-6

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

ROOF SUBSIDENCE LEVELS
2 SOUTH

PA78-040

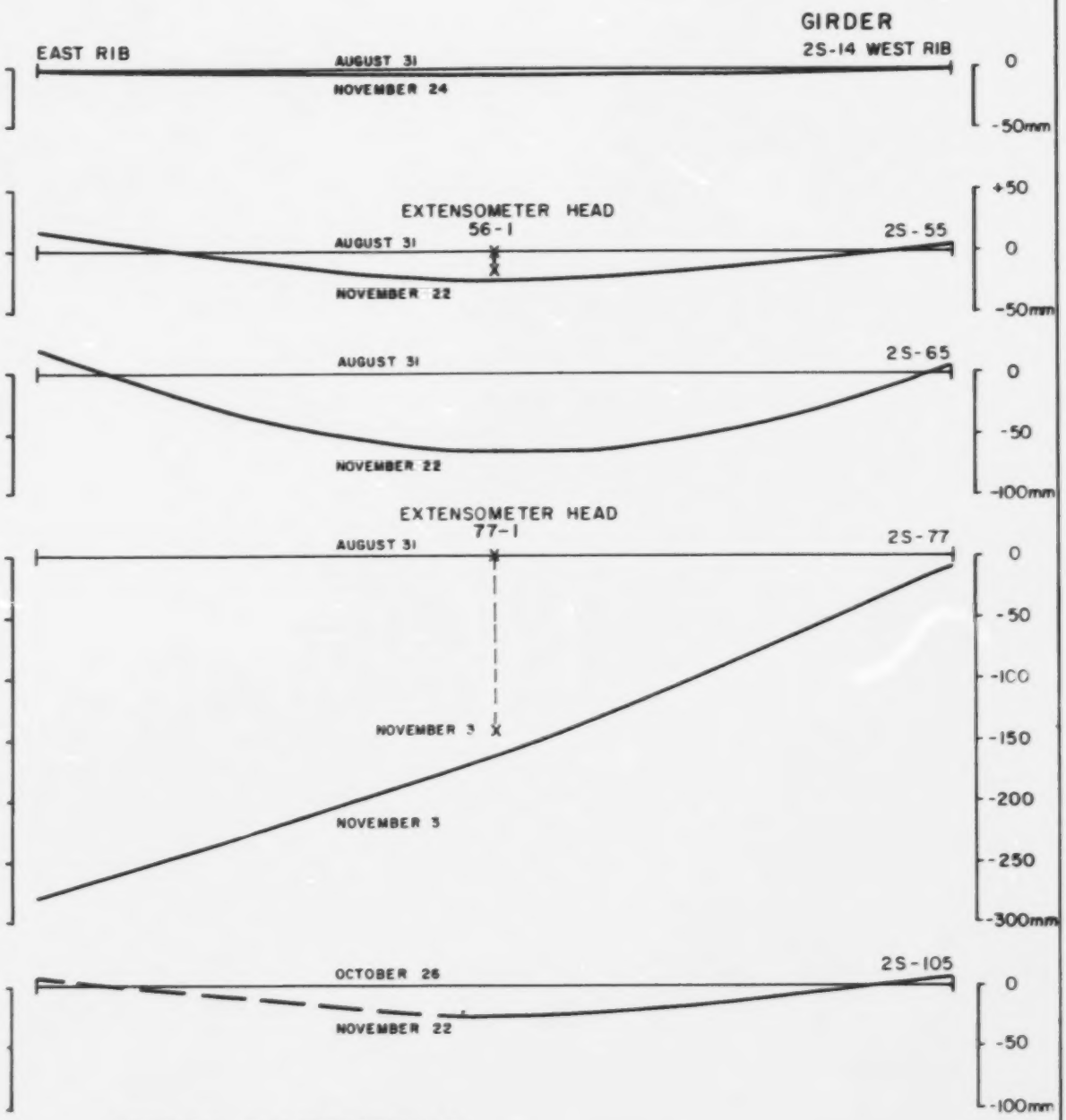


FIGURE 4-7

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

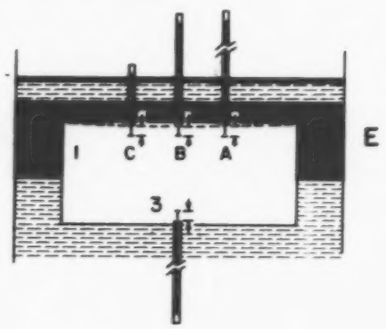
TRIAL MINING PROGRAM – KIPP

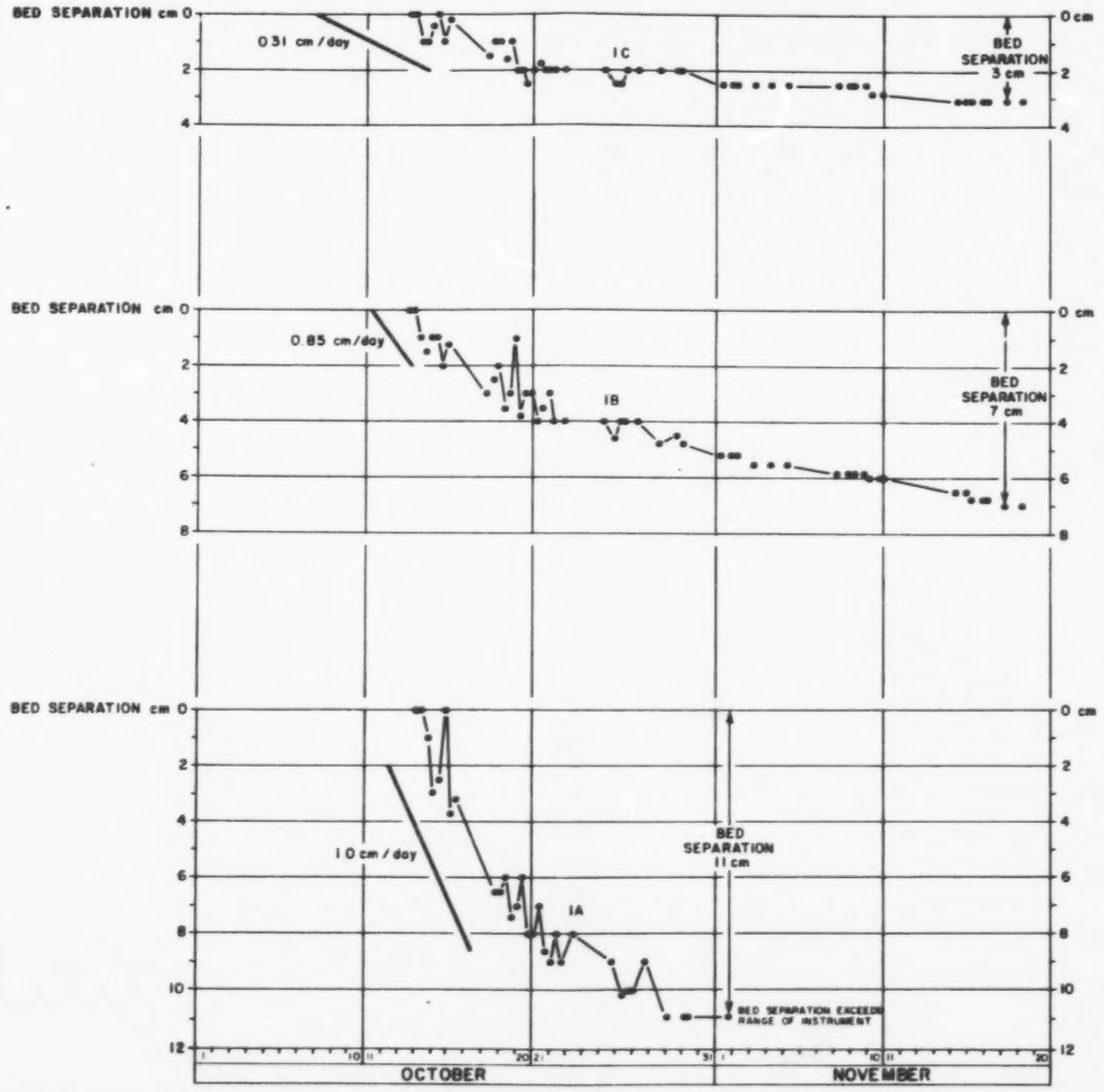
GIRDER AND EXTENSOMETER DISPLACEMENT

2 SOUTH

PA 73-039

ROOF EXTENSOMETER





IS-120-1C (1.2 m)

IS-120-1B (2.4 m)

IS-120-1A (3.6 m)

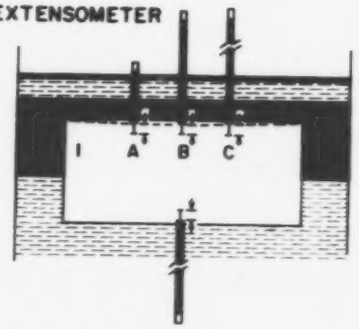
FIGURE 4-8

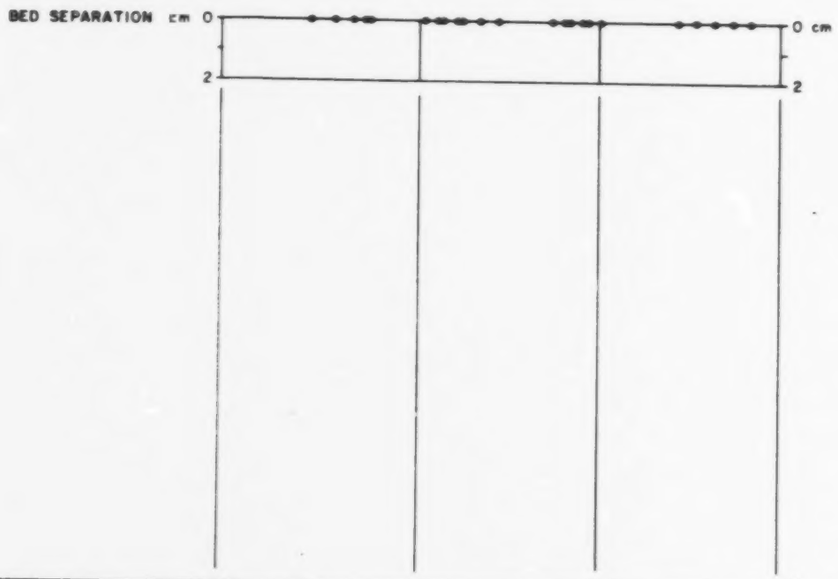
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
ROOF EXTENSOMETERS
IS-120-1

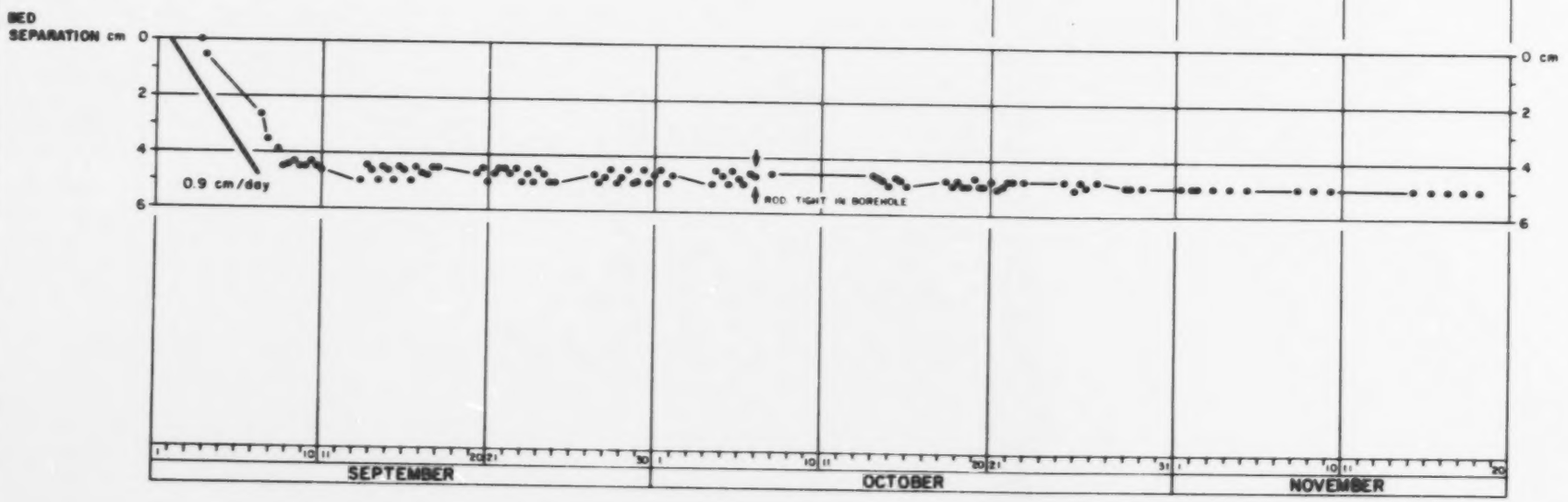
PA73-043

ROOF EXTENSOMETER





4W-20-IC (3.6m)



IS-53-I (2.4 m)

FIGURE 4-9

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
ROOF EXTENSOMETERS
4W-20-I, IS-53-I

PA73-042

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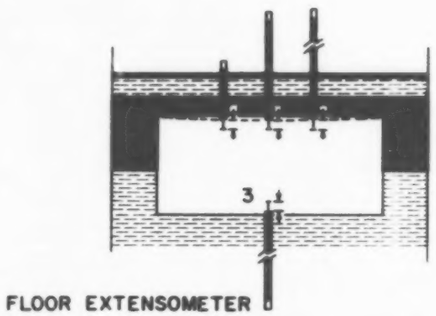

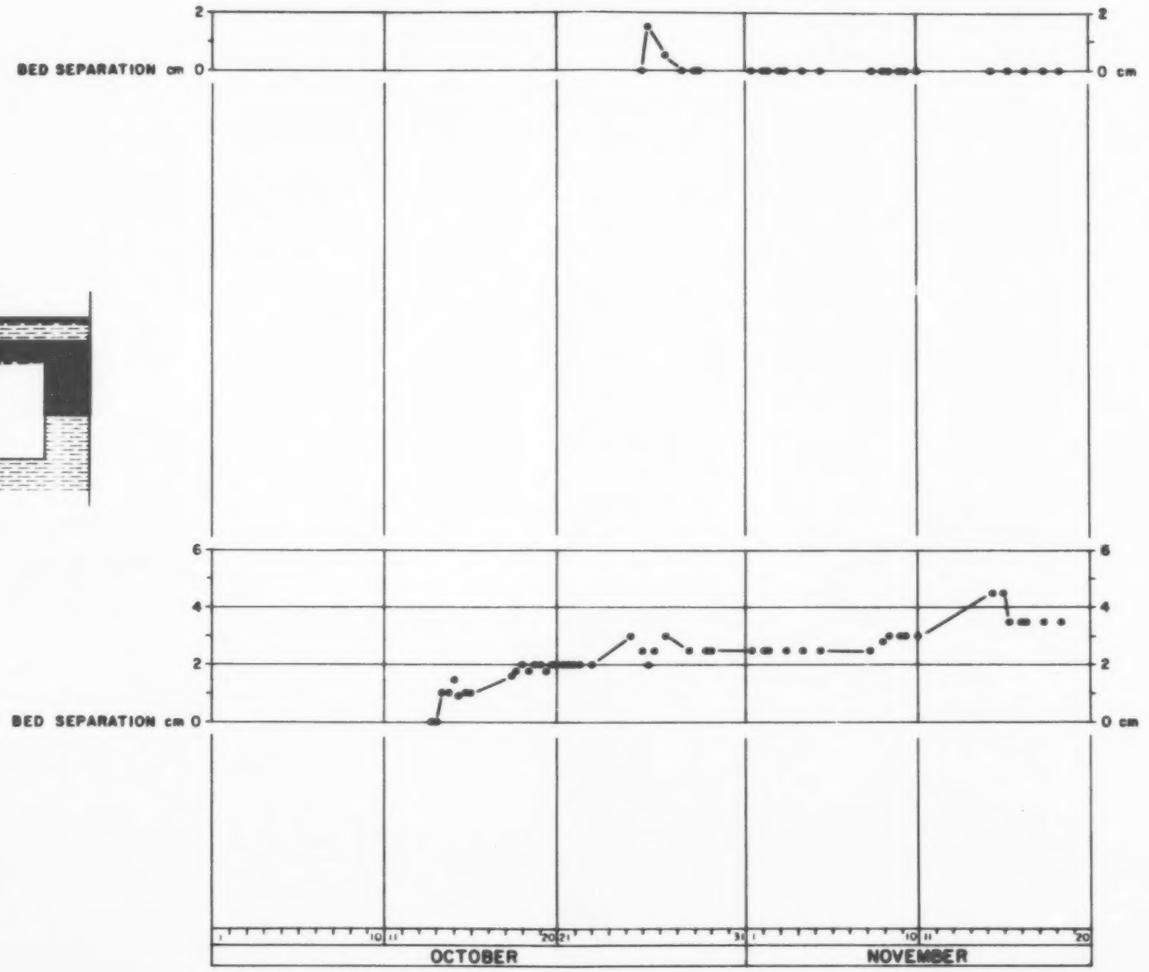

4W-20-3

IS-120-3

FIGURE 4-10

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
FLOOR EXTENSOMETERS
4W-20-3, IS-120-3

PA73-044

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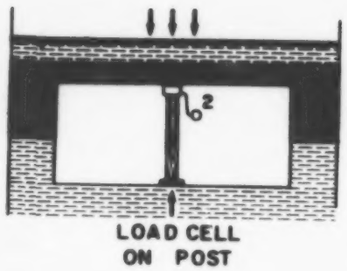
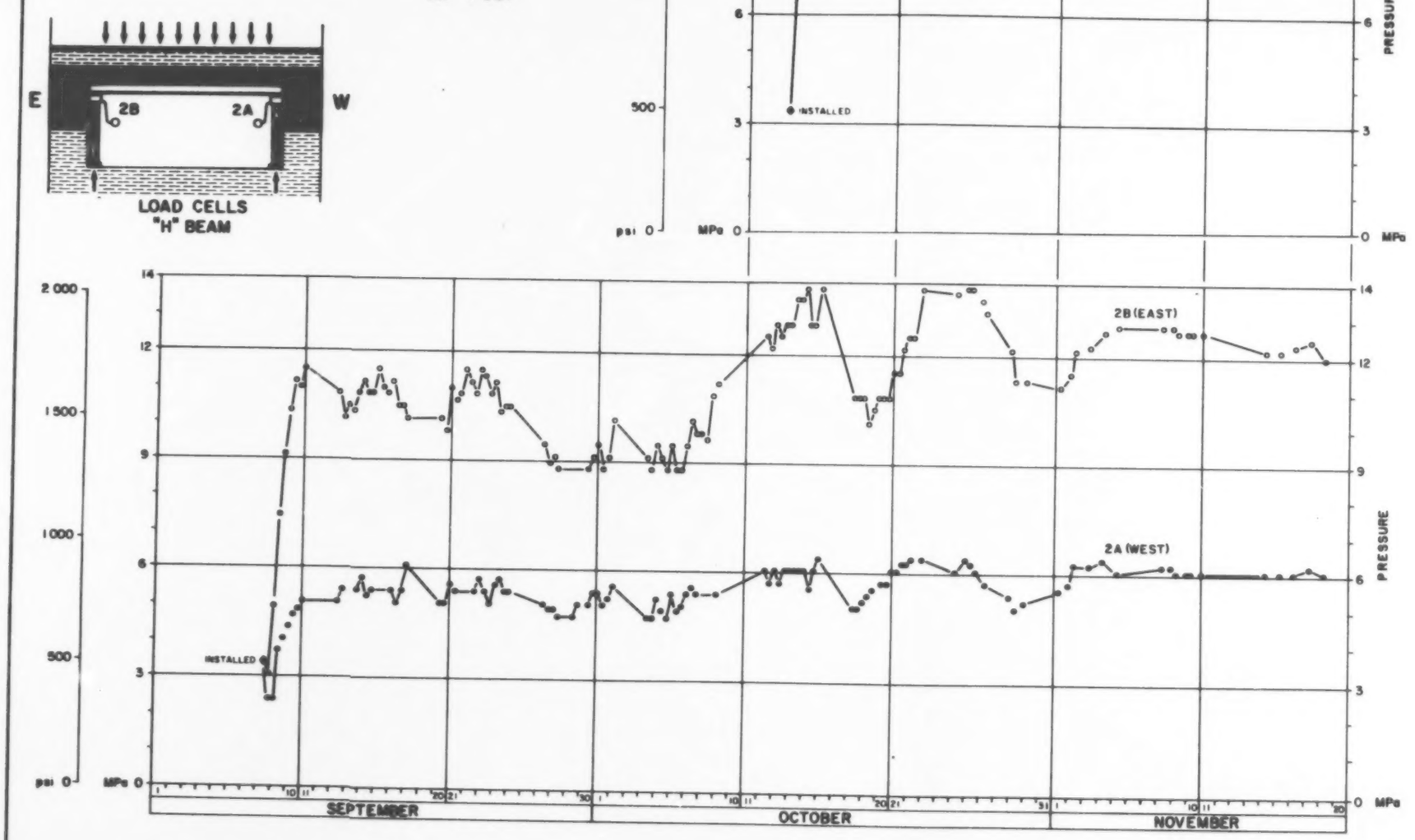
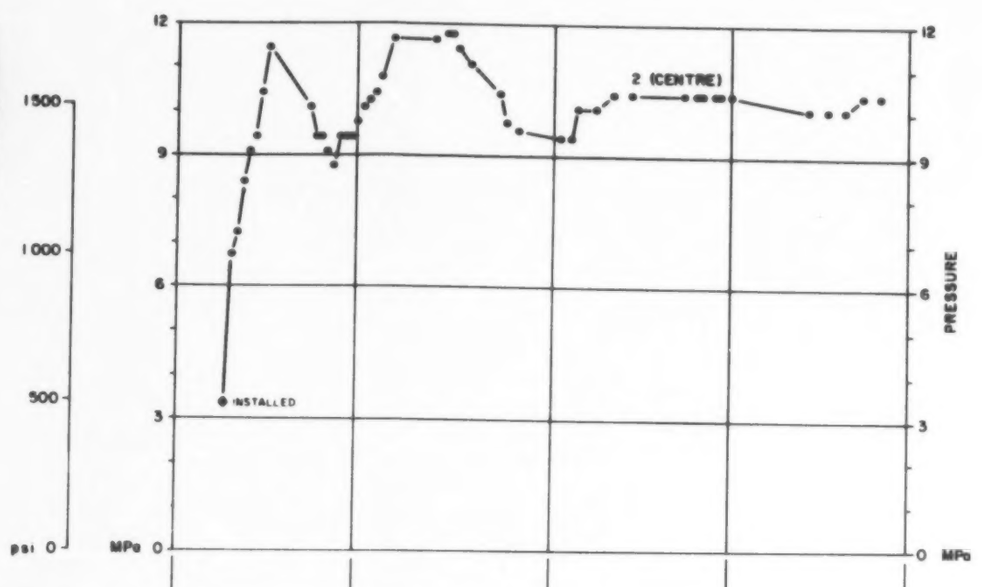
Golder Associates







IS-124-2

IS-67-2A, 2B

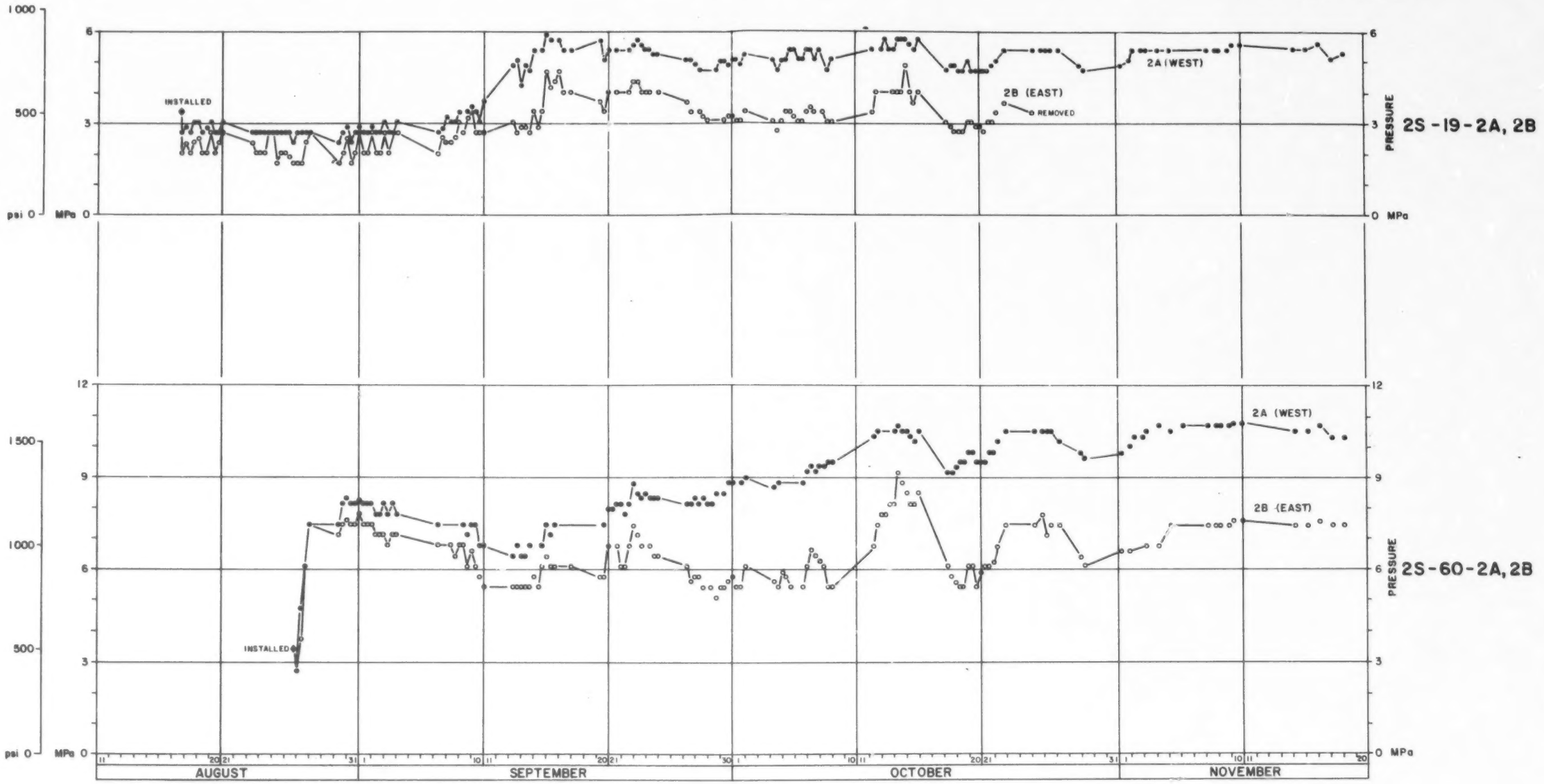
FIGURE 4-11

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
LOAD CELLS
IS-124-2, IS-67-2

PA73-048

Associated Mining Consultants Ltd.
Golder Associates



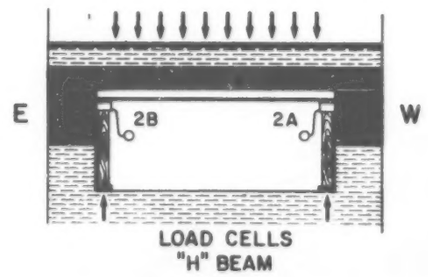
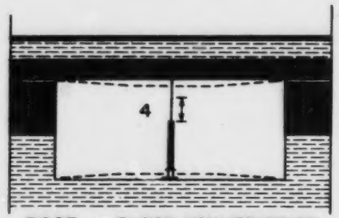


FIGURE 4-12

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
LOAD CELLS
2S-19-2, 2S-60-2

PA73-045



ROOF – FLOOR CONVERGENCE

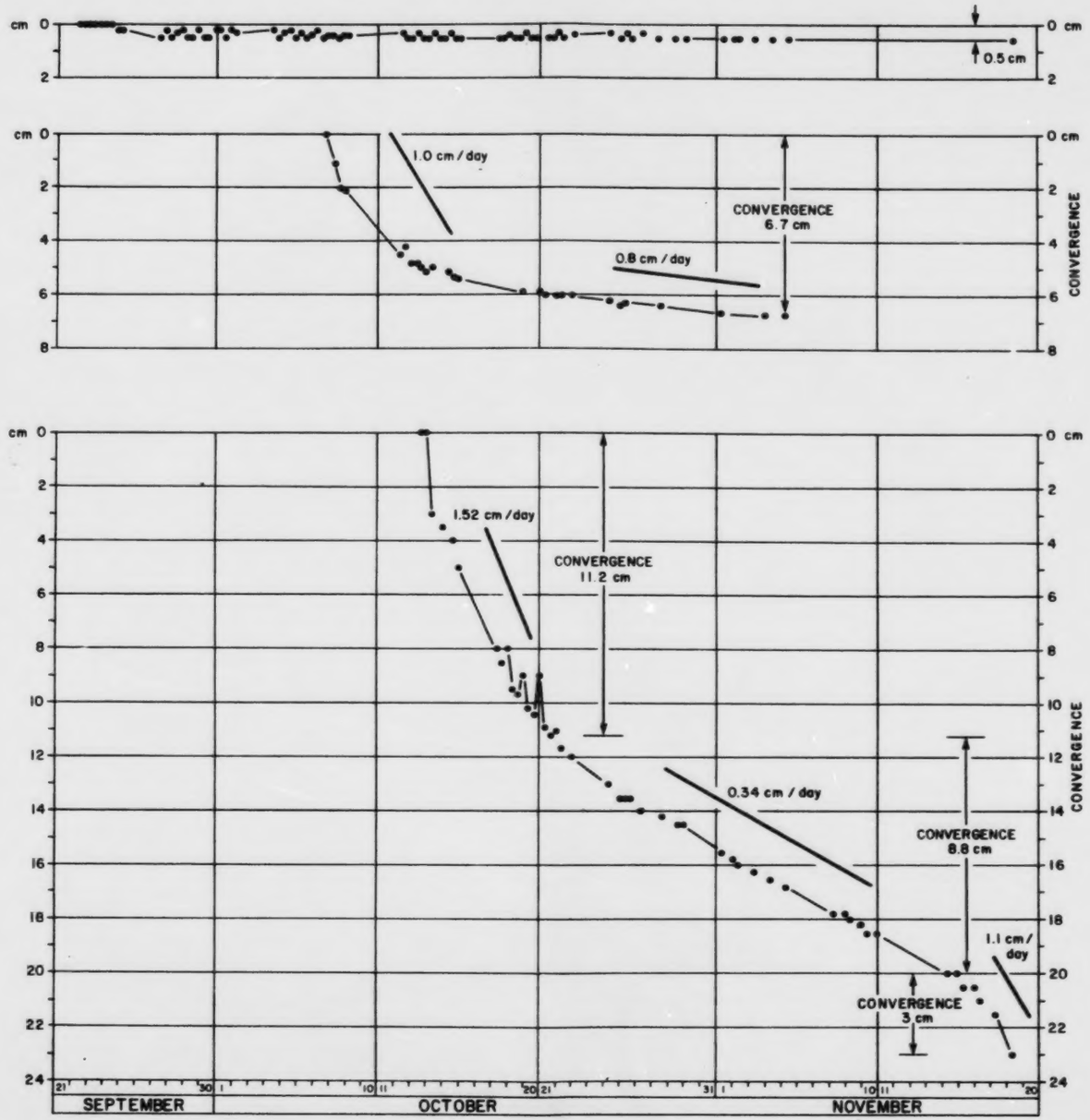


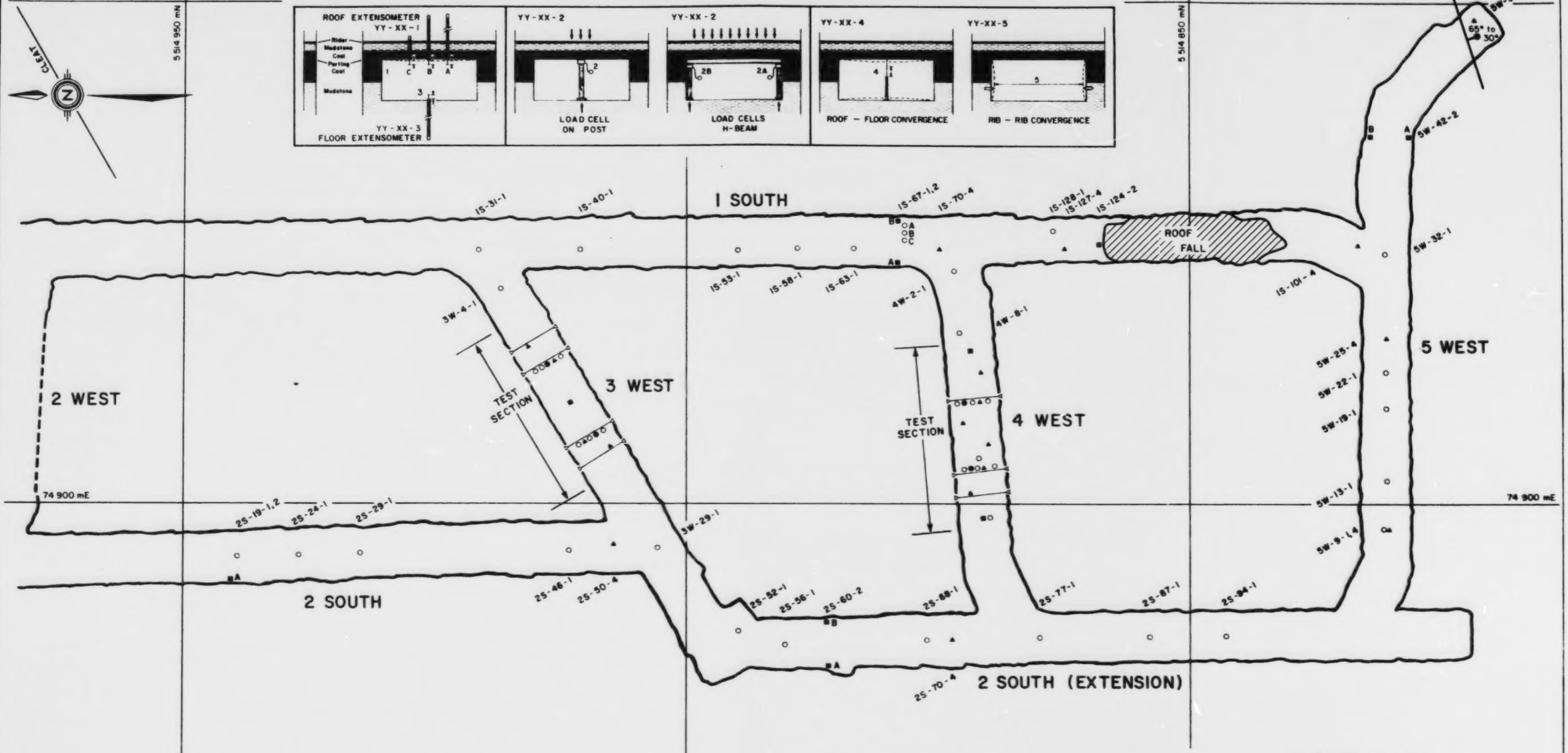
FIGURE 4-13

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP
INSTRUMENTATION GRAPHS
CONVERGENCE RODS
3W-9-4, 2S-70-4, 1S-120-4

PAT3-046

Associated Mining Consultants Ltd.
Golder Associates



INSTRUMENTATION

SYMBOL	NAME	CODE
○	ROOF EXTENSOMETER	YY-XX-1
■	LOAD CELL	YY-XX-2
●	FLOOR EXTENSOMETER	YY-XX-3
▲	CONVERGENCE ROD	YY-XX-4
⊢⊣	RIB - RIB CONVERGENCE	YY-XX-5

YY - DENOTES ROADWAY No.
XX - DENOTES GIRDER No.

FIGURE 4-14

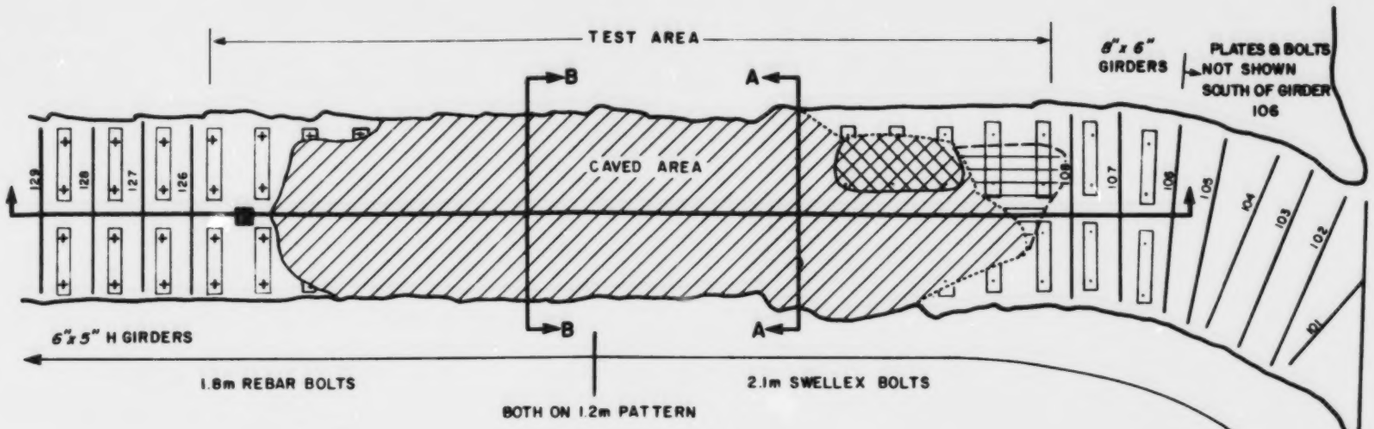
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

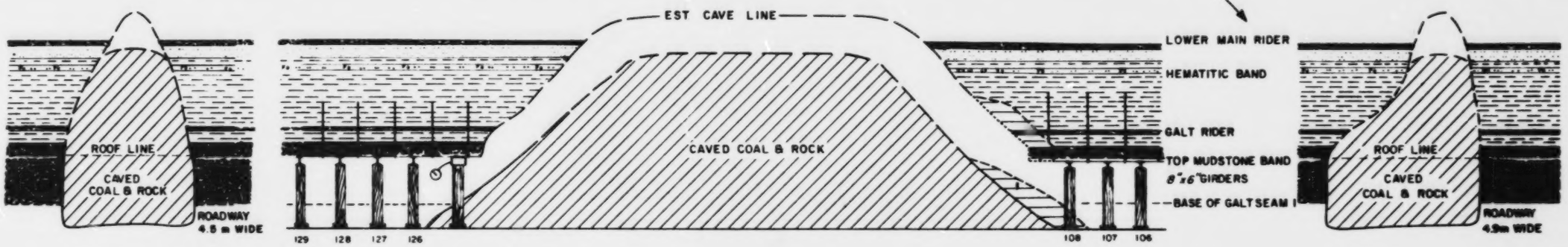
TRIAL MINING PROGRAM – KIPP

FINAL INSTRUMENTATION
NOVEMBER 23 / 1982

PA73-017A

PLAN VIEW





SECTION B-B

LONGITUDINAL SECTION

SECTION A-A

FALL NOV.22, 1982

LIMIT OF FALL NOV 23, 1982

FALL DEC.7, 1982

0 2 4 6 8 10 12 14 m

FIGURE 4-15

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

I SOUTH ROOF FALL

PA75-089

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SECTION 5 - GEOTECHNICAL DRILLING AND TESTING

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.....Cont'd

PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP

SECTION 5 - GEOTECHNICAL DRILLING AND TESTING

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SECTION 5 - GEOTECHNICAL DRILLING AND TESTING

5.1 GEOTECHNICAL PROGRAM

The underground geotechnical program at Kipp was carried out between August 7 and November 28, 1982 and was co-ordinated with the TMP. The geotechnical program consisted of the following:

- core drilling
- core logging
- selected sampling
- in situ strength testing
- laboratory testing

5.1.1 Core Drilling

Four holes were drilled at two locations, as shown in Figure 3-1. At each location holes were drilled into both the floor and roof strata. Drilling was carried out using a pneumatically powered screw-fed diamond drill, with a double-tube core barrel and water flush to obtain NX core. All drilling was carried out in the presence of a Golder Associates' geologist.

5.1.2 Core Logging

All core was carefully extruded and logged immediately on removal from the core barrel in order to record its condition in an undisturbed state prior to handling and transportation. Selected samples were wrapped in polythene and silver foil to prevent moisture loss prior to laboratory testing.

The core was subsequently taken to Golder Associates Calgary laboratory for more detailed inspection and core photography.

All drill core was photographed to provide a permanent visual record of ground conditions and the strata encountered above and below the Galt Seam at Kipp. Core photographs are shown in Figures 5-1, 5-2, 5-3 and 5-4.

5.1.3 Selected Sampling

Representative core samples of the various lithological units encountered in the drillholes were selected for specific laboratory testing. In addition, bulk samples of the seam, and exposed roof and floor strata were taken for additional laboratory testing.

A complete list of sample locations and tests carried out is shown in Tables 5-1 and 5-2.

5.1.4 In Situ Strength Testing

The Schmidt Hammer index test was performed on random exposures, and was used only as a rapid indicator of spot strengths of the rock mass during sample selection for more reliable rock strength testing.

5.1.5 Laboratory Testing

Laboratory testing was carried out in both Golder Associates Calgary and Vancouver laboratories. Specialized spectrographic analysis on 4 samples was performed by Geotex Consultants Ltd. of Vancouver, a certified chemical laboratory.

A total of 103 samples were subjected to a series of laboratory tests. Tests carried out were as follows:

- unconfined compressive strength
- point load index
- slake durability
- free swell
- bulk density
- calcium content
- natural moisture content
- Atterberg limits

In addition, 13 point load tests were performed on bulk samples in the mine. The test results are discussed in Section 5.3.2.

5.2 ROCK QUALITY

The following parameters were recorded in order to assess and systematically compare to the parameters determined by previous geotechnical programs:

- hardness
- bedding break intervals
- rock quality designation (RQD)

5.2.1 Hardness

Lithological boundaries and zones of weakness were identified in drill core by systematically scratching the core with a knife as a rapid and simple means of identifying the relative hardness of the various lithological units, arbitrarily classifying the most prominent mudstones as medium hard.

5.2.2 Bedding Break Intervals

Spacing measurements were carried out when the core was first exposed, and the bedding break spacings are shown on the geotechnical core logs (Figures 5-5 and 5-6). The logs should be used with reference to Table 5-3 which shows Deere's classification system for discontinuity spacing.

This parameter is highly subjective but a few general conclusions can be drawn:

- The bedding break spacing for the mudstones range widely, from 5 cm to more than 1 m, but is generally between 0.1 m to 0.5 m close to moderately close.
- Bedding break spacing for the siltstones is generally moderately close to wide, ranging from 0.5 m to more than 1 m.
- Bedding break spacing for the sandstone is generally close to moderately close, ranging from 0.1 m to 0.5 m.

These general conclusions are inconsistent with the previously obtained results in the area where values ranging from 4 cm to 0.1 m were recorded. This could result from the different drilling procedure, diamond drilling (as performed in the mine) tending to produce less disturbed core than drilling from surface with tungsten carbide bits (as performed in 1978).

5.2.3 Rock Quality Designation (RQD)

Rock Quality Designation (RQD), is the cumulative axial length of intact rock which occurs in pieces longer than 0.1 m (unbroken by natural discontinuities) expressed as a percentage of the total core run, or of a nominal interval such as 1 m.

Rock quality designation measurements made over each run of core, or arbitrarily between certain lithologic boundaries, were recorded and are shown graphically on the logs. These should be used with reference to Table 5-4.

In general terms, RQD values for the roof strata approach 100% within 2 m of the top of the Galt Seam. The first 2.5 m of the roof strata in BH 82-4 appeared highly broken. This could be

related to the plate load testing performed by CMRC in the immediate vicinity of this drillhole or a reflection of the very poor roof conditions encountered when this area was mined. Dames and Moore recorded values ranging from almost 60% near the Galt Seam to approximately 80% nearer the Rider Seam. The higher RQD values recorded in the 1982 program probably reflect the reduced disturbance of the core by the drilling procedure used underground.

RQD values recorded for the immediate floor of the Galt Seam ranged from 15% to 55% for the first 1.5 m. These values increased with depth to between 60% and 80%. These are in general agreement with values recorded by Dames and Moore throughout the Kipp area, where values recorded previously ranged between 55% and 85%.

5.3 LABORATORY TEST RESULTS

5.3.1 Uniaxial Compression Testing

Uniaxial compression testing was performed on 21 samples. The uniaxial compression test is the basic strength index test for rock strength classification. The test is easily performed in the laboratory on core samples. Compression testing was also carried out on bulk samples by trimming to rectangular prisms, and applying the compressive force perpendicular to the stratification. The results of these tests are presented in Table 5-5. These should be used with reference to Table 5-6.

Two tests were performed on Galt coal samples, indicating a strength of approximately 10 MPa. This is consistent with the mean strength of 9 MPa obtained by Dames and Moore for coal samples obtained from Holes 78-1 and 78-2 in the Kipp area.

The Dames and Moore 1979 Second Stage Geotechnical Study (BH 78-3, 78-4, 78-5, 78-6 and 78-7) recorded a mean strength of 32 MPa for mudstones from both roof and floor strata, although no distinction was made between mudstone samples from the roof or floor strata. Laboratory testing carried out in 1982 indicates that the mudstones in the roof strata have an average strength value of approximately 50 MPa. Similar mudstones encountered in the floor sequence recorded lower strength values, with an average value of 34 MPa. Both mudstone sequences can be classified as moderately strong rocks.

No significant difference in strength was observed between the mudstones and carbonaceous mudstones in the seam floor sequence.

The strongest lithological unit in the floor sequence is the silty sandstone encountered in BH 82-1 (2.6 m - 2.7 m below seam floor). The strength value obtained was 54.4 MPa. Dames and Moore recorded lower average values for these sandstones of between 21 MPa and 33 MPa.

The sandstones above the Rider Seam at the top of the Oldman Formation, have an unconfined compressive strength of approximately 35 MPa which is lower than that of the roof mudstones. Visual inspection of the core indicated that its tuffaceous matrix is highly susceptible to alteration.

Only one test was performed on the ferruginous siltstone encountered in the roof sequence. The strength value obtained (82.1 MPa) confirms the visual interpretation that these are the strongest units in the sequence.

The strength of the floor siltstones appears to be intermediate between the strength of mudstones and floor sandstones. Strength obtained in two tests were approximately 40 MPa.

5.3.2 Point Load Index

Point load tests were performed on 35 samples. The point load test is a relatively fast and inexpensive procedure to obtain approximate unconfined compressive strength parameters of intact rock whereby samples are subjected to a tensile stress induced between two hardened steel points. The unconfined compressive strength is then empirically related to the point load index obtained from the test.

Point load testing on bulk samples was carried out with a portable testing unit. Because of the difficulty in preparing samples due to the physical properties of the strata (i.e. fissility of shales/-mudstones) more emphasis was placed on controlled qualitative laboratory testing.

The majority of samples (29) were loaded perpendicular to the stratification in order to determine rock material strength. Six samples were loaded parallel to the stratification to obtain an approximate value for the load strength along bedding planes. The results of these tests are presented in Table 5-5.

For coal samples there is a high degree of consistency between the point load strength and the uniaxial compressive strength values. Furthermore, no apparent strength anisotropy exists, as samples loaded parallel to the bedding recorded similar strength values as those obtained for tests perpendicular to the bedding.

Point load test results for mudstones from both the roof and floor sequence have a far greater scatter than the uniaxial compression test results, and are generally higher. The following statistical parameters were derived from the point load test results:

- roof mudstones (10 tests) - mean: 51 MPa
  standard deviation: 30 MPa
- floor mudstones (9 tests) - mean: 59 MPa
  standard deviation: 18 MPa

The point load test results for roof mudstones confirms the results obtained from uniaxial compression tests. However, the results from floor mudstone samples from the two testing methods are not consistent. Because of the larger scatter of point load test results, and the limited number of tests carried out, more emphasis for preliminary design parameters should be placed on the results of uniaxial compression tests.

5.3.3 Slake Durability

The slake durability test is designed to estimate the resistance of argillaceous rocks in particular to a combination of abrasion and moisture exposure. The slake durability index is obtained by rotating a standard sample of oven-dried material in a drum immersed in water. The drum is made of a standard mesh and material that breaks down under the tumbling action passes through and is retained, oven-dried, and weighed. The slake durability index (first cycle) is then calculated as the percentage ratio of final to initial dry sample weight. This procedure is repeated to obtain the slake durability index - second cycle, which is generally recommended.

Slake durability tests were performed on 20 samples to indicate the degree of disintegration that could result within the different strata units during mining operations. The results of these tests are shown in Table 5-7 while typical examples are shown photographically in Figures 5-7, 5-8, 5-9 and 5-10.

The slake durability indices for the Galt coal (89.9% and 64.3%) are consistent with the results obtained by Dames and Moore for samples from BH 78-1 and BH 78-2, (90% and 70%). These results indicate a relatively high resistance to mechanical breakdown, and this is illustrated photographically in Figure 5-7. Only the test results from samples taken from BH 78-1 and BH 78-2 were compared with the 1982 test program results as Dames and Moore stated that some samples from the late 1978 geotechnical program had physically broken down prior to testing, and would have probably resulted in artificially high slake durability indices.

Testing on mudstone samples from the immediate floor (0 m to 0.4 m below the base of the Galt Seam) recorded values varying from 87.1% to 65.2% for the first cycle, and 52.7% to 20.1% for the second cycle. Average values obtained by Dames and Moore were 60% and 13% for the first and second cycle respectively, and these results are consistent with the results of the 1982 program.

5.3.4 Free Swell

A total of 9 mudstone samples and 1 siltstone sample were allowed to swell freely in a consolidation machine under atmospheric pressure. The values indicating the relative volume change are shown in Table 5-8.

In most tests the relative volume changes reached their asymptotic maximum within 2000 minutes (33 hours 20 minutes).

Generally the swelling potential for the siltstones and mudstones is very low (1.2% to 1.7%). The high value (4.8%) obtained in the roof sample at 4 to 4.2 m above the seam in BH 82-2 probably results from the presence of montmorillonite clay minerals that were found to exist in these strata.

The swelling potential for the immediate floor strata is substantially higher (between 3.9% and 11.6%) and confirms the higher plasticity index.

5.3.5 Calcite Content

The results of the 4 determinations of the calcite content in typical floor strata are shown in Table 5-7 and are all below 1.5%.

5.3.6 Atterberg Limits

Atterberg limits were determined on 6 samples. The results are shown in Table 5-7. The plasticity of the mudstones in the floor sequence is slightly higher than that of the roof mudstones, although no significant difference was recognized during visual inspection. The lowest recorded plastic limit, being the critical moisture content above which a clay reverts to the plastic state, was 23.4%. This is much higher than the in situ natural moisture content of 9% to 10%. (Section 5.5 g), and indicates that, unless extremely wet conditions were to be artificially induced during mining operations, the floor strata would not become 'slick'.

5.3.7 Moisture Content

As shown in Table 5-5, the moisture contents for the carbonaceous mudstones and mudstones of the immediate floor strata generally range between 5.5% and 9.7%. These results are consistent with test samples from BH 78-1 and BH 78-2 (Dames and Moore) which recorded average natural moisture content values of 9.3%.

5.3.8 Bulk Density

Bulk densities determined for 10 samples are shown in Table 5-9.

Mudstones in the immediate floor strata have densities varying between 1.88 and 2.23. The carbonaceous mudstone that occurs as a thin band immediately at the base of the Galt Seam has a bulk density of 1.58.

The bulk density of the siltstone and sandstone samples tested were 2.4 and 2.5 respectively.

5.3.9 Petrographic and X-Ray Diffraction Analyses

The presence or otherwise of expanding clay minerals could have significant implications for both roof and floor stability. Mineralogical identification of quartz and clay minerals was achieved qualitatively by x-ray diffraction supplemented by thin section observations under the microscope. These analyses were carried out by Geotex Consultants Limited.

Ten samples of drill core from the underground drilling program were examined by means of x-ray diffraction of which 7 were

examined petrographically. The location of samples tested and their mineralogy are shown in Table 5-10.

The x-ray diffractograms of samples show that all but Sample SA1 contain major amounts of clay minerals of which montmorillonite is dominant. Sample SA5 is more than 95% montmorillonite with no other detectable 'clay minerals' or quartz, and explains the rapid dessication of the thin parting in the Galt Seam on exposure during test mining.

TABLE 5-1
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

LIST OF GRAB SAMPLES SELECTED AND TESTED

SAMPLE #	LOCATION	DISTANCE ABOVE (+) OR BELOW (-) GALT (m) From	To	GEOTECHNICAL UNIT	LITHOLOGICAL DESCRIPTION	TESTS CARRIED OUT
1	2S-113	+0.9		Roof	mudstone	FS UC PL BD
2	2S-112	+0.54	+0.79	Roof	mudstone	FS At UC PL BD
5	2S-113			Galt Upper Parting	bentonitic mudstone	MC
7	3W-40			Galt Upper Parting	bentonitic mudstone	SD MC
8	3W-40			Galt	coal	UC SD PL
14	2S-112	-0.00	-0.20	Floor	mudstone	MC**
15	3W-40	-0.00	-0.19	Floor	silty mudstone	FS UC SD MC PL
18	1S-76	-0.33	-0.48	Floor	silty mudstone	FS UC SD MC PL BD
19	2S-27	-0.05	-0.25	Floor	silty mudstone	FS UC SD MC PL BD
24	3W-40	-0.00	-0.19	Floor	silty mudstone	FS UC SD MC BD
27	2S-112	-0.00	-0.10	Floor	mudstone	MC*
28	2S-112	-0.32	-0.45	Floor	mudstone	At SD MC PL Chem.
34	1W-42	+0.20		Roof	mudstone	At UC PL
38	1S-76	-0.05	-0.27	Floor	carbonaceous silty mudstone	At UC SD PL Chem.
39	1S-76	-0.27	-0.35	Floor	silty mudstone	At UC SD PL Chem.
41	2S-112	-0.00	-0.00	Floor	carbonaceous mudstone	At UC SD MC BD Chem.
60	2S-120 approx.	-0.20	-0.40	Floor	mudstone	SD
61	2S-120 approx.	-0.20	-0.40	Floor	mudstone	SD
62	2S-51	-0.30		Floor	mudstone	SD
63	2S-113	-0.00	-0.20	Floor	mudstone	SD
64	2S-1			Galt	coal	PL
65	2S-1			Galt	coal	PL
66	2S-1	+0.00	+0.20	Roof	mudstone	PL
67	2S-1	+0.00	+0.20	Roof	mudstone	PL
68	2S-1	+0.00	+0.20	Roof	mudstone	PL
69	2S-1	+0.00	+0.20	Roof	mudstone	PL
70	2S-1	+0.00	+0.20	Roof	mudstone	PL
71	2S-1	+0.00	+0.20	Roof	mudstone	PL
72	2S-1	+0.00	+0.20	Roof	mudstone	PL
73	2S-1	+0.00	+0.20	Roof	mudstone	PL
74	1S-76	-0.00	-0.20	Floor	mudstone	PL
75	1S-76	-0.00	-0.20	Floor	mudstone	PL
76	1S-76	-0.00	-0.20	Floor	mudstone	PL

LEGEND

FS	Free Swell
UC	Uniaxial Compression Test
PL	Point Load
BD	Bulk Density
SD	Slake Durability
MC	Moisture Content
At	Atterberg Limits
Chem.	Chemical Analysis
*	Before CMRC Testing
**	After CMRC Testing

TABLE 5-2
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

LIST OF CORE SAMPLES SELECTED AND TESTED

DRILLHOLE No.	DEPTH /HEIGHT (m)	GEOTECH-NICAL UNIT	LITHOLOGICAL DESCRIPTION	TESTS CARRIED OUT
82-1	0.9-1.0	Floor	carbon bearing mudstone	SD
	2.2-2.3	Floor	bentonite bearing silty sandstone	UC
82-2	2.6-2.7	Roof	mudstone with arenaceous laminae	UC BD
	4.0-4.2	Roof	mudstone with arenaceous laminae	FS UC
	7.5-7.6	Roof	wavy bedded sandstone	UC BD
82-3	0.8-0.85	Floor	Mudstone	SD
	3.6-3.75	Floor	siltstone	UC BD
	9.2-9.35	Floor	siltstone	UC BD
82-4	3.1-3.3	Roof	laminated siltstone	FS UC BD
	4.0-4.7	Roof	massive mudstone	FS BD

LEGEND

FS	Free Swell
UC	Uniaxial Compression Test
BD	Bulk Density
SD	Slake Durability

TABLE 5-3
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

DEERE'S CLASSIFICATION FOR JOINT/FRACTURE AND BEDDING BREAK SPACING

DESCRIPTION	SPACING OF JOINTS	ROCK MASS GRADING
Very Wide	3 m	Solid
Wide	1 m to 3 m	Massive
Moderately Close	0.3 m to 1 m	Blocky/Seamy
Close	50 mm to 300 mm	Fractured
Very Close	50 mm	Crushed and Shattered

TABLE 5-4

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP MINE

DEERE'S CLASSIFICATION FOR ROCK QUALITY DESIGNATION (RQD)

RQD	ROCK QUALITY
25%	Very Poor
25 - 50%	Poor
50 - 75%	Fair
75 - 90%	Good
90 - 100%	Very Good

TABLE 5-5

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP MINE

TEST RESULTS UNIAXIAL COMPRESSIVE STRENGTH AND POINT LOAD

SAMPLE #	LOCATION	DRILLHOLE #	DISTANCE ABOVE (+) OR BELOW (-) GALT (m) From	To	UNIAXIAL COMPRESSIVE STRENGTH (MPa)	POINT LOAD STRENGTH PERPENDICULAR TO STRATIFICATION	INDEX (MPa) PARALLEL TO STRATIFICATION
Galt Coal							
8	3W-40				8.8/11.2		
64	2S-1						
65	2S-1					7.5	3.5/8.6/7.5
Mudstone (Roof)							
66	2S-1		+0.00	+0.20		(22.7)	
67	2S-1		+0.00	+0.20		(16.8)	
68	2S-1		+0.00	+0.20		47.5	
69	2S-1		+0.00	+0.20		39.6	
70	2S-1		+0.00	+0.20		34.0	
71	2S-1		+0.00	+0.20		77.3	
72	2S-1		+0.00	+0.20		30.6	
73	2S-1		+0.00	+0.20		37.8	
34	1W-42		+0.2		41.6	54.2	
						114.2	
						82.6	
		82-2	+2.6	+2.7	55.9		
		82-2	+0.40	+4.4	53.0		
Siltstone (Roof)							
		82-4	+3.1	+3.3	82.1		
Sandstone (Roof)							
		82-2	+7.5	+7.6	35.5		
Carbonaceous Mudstone (Floor)							
41	2S-112		-0.00	-0.10	30.5		
38	1S-76		-0.05	-0.27	33.1	65.0	
	1S-76		-0.05	-0.27		50.2	
Mudstone (Floor)							
15	3W-40		-0.00	-0.19	32.2	31.3	
24	3W-40		-0.00	-0.19	43.9		
74	1S-76		-0.00	-0.20			7.0
75	1S-76						11.6
76	1S-76						11.1
28	2S-112		-0.32	-0.45		56.4	
18	1S-76		-0.33	-0.48	40.1	(24.2)	
						37.9	
						45.6	
Mudstone (Floor) Cont'd)							
19	2S-27		-0.05	-0.25	30.6	80.4	
	2S-27		-0.05	-0.25		(17.4)	
	2S-27		-0.05	-0.25		67.9	
	2S-27		-0.05	-0.25		64.8	
25	2S-27		-0.25	-0.40	27.4		
39	1S-76		-0.27	-0.35	30.1	61.7	
	1S-76		-0.27	-0.35		81.1	
	1S-76		-0.27	-0.35		(101.8)	
2	2S-112		+0.54	+0.79	(24.8)	86.9	
1	2S-113		+0.9		51.6	(161.0)	
						61.2	
Silty Sandstone (Floor)							
		82-1	-2.2	-2.3	54.4		
Siltstone (Floor)							
		82-3	-3.60	-3.75	41.1		
			-9.20	-9.35	42.0		

NOTE Bracketed values are anomalous and were not used in engineering considerations

TABLE 5-6
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

GOLDER ASSOCIATES STRENGTH CLASSIFICATION

TERM	U.C.S. (MPa)
Very Strong	100 - 200
Strong	50 - 100
Moderately Strong	12.50 - 50.00
Moderately Weak	5.00 - 12.50
Weak	1.25 - 5.00
Very Weak	0.60 - 1.25

TABLE 5-7

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP MINE

TEST RESULTS - SLAKE DURABILITY, CALCITE CONTENT, MOISTURE CONTENT, AND ATTERBERG LIMITS

SAMPLE #	LOCATION	DRILL HOLE #	DISTANCE ABOVE (+) OR BELOW (-) GALT (m) From	To	SLAKE DURABILITY (% RETAINED) 1st CYCLE	2nd CYCLE	CALCITE CONT. (%)	MOISTURE CONT. (%)	ATTERBERG LIMITS LIQUID (%)	PLASTIC (%)	PLAST. INDEX (%)
Galt (Coal)											
8					89.9	64.3					
Galt (Upper Parting)											
7	3W-40				66.3	38.5		14.7			
5	2S-113							17.4			
Mudstone (Roof)											
34	1W-42		+0.20								
2	2S-112		+0.54	+0.79					48.3	25.6	22.7
									36.9	19.9	17.0
Carbonaceous Mudstone (Floor)											
41	2S-112		-0.00	-0.10	97.2	56.0	0.47	8.6	38.0	29.5	8.5
38	1S-76		-0.05	-0.27	85.1	57.4	0.58		54.0	25.2	28.8
Mudstone (Floor)											
27	2S-112		-0.10	-0.20				9.7			
15	3W-40		-0.00	-0.19	83.3	39.2		8.3			
24	3W-40		-0.00	-0.19	84.6	52.7		7.6			
14	2S-112		-0.00	-0.20				9.1			
63	2S-113		-0.00	-0.20	29.7						
19	2S-27		-0.05	-0.25	79.3	44.3		5.5			
60	2S-120 approx.		-0.20	-0.40	86.4	51.9					
61	2S-120 approx.		-0.20	-0.40	87.1	20.1					
25	2S-27		-0.25	-0.40	80.4	51.6		6.4			
39	1S-76		-0.27	-0.35	74.6	29.0	1.31		54.0	23.4	30.6
62	2S-51		-0.30		65.2						
28	2S-112		-0.32	-0.45	72.8	29.8	0.61	7.9	55.5	24.5	31.0
18	1S-76		-0.33	-0.48	69.0	22.8		6.3			
		82.3	-0.80	-0.85	96.4	92.4					
		82-1	-0.90	-1.00	96.2	92.2					
		82-1	-2.20	-2.30	94.7	92.6					
		82-3	-3.60	-3.75	94.9	90.4					
		82-3	-9.10	-9.35	94.1	89.3					

TABLE 5-8
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

TEST RESULTS - FREE SWELL

SAMPLE #	LOCATION	DRILLHOLE #	DISTANCE BELOW (-) OR ABOVE (+) GALT (m) From	To	FREE SWELL %
Mudstone (Roof)					
1	2S-113		+0.9		1.7
2	2S-112		+0.54	+0.79	1.4
		82-2	+4.0	+4.2	4.8
		82-4	+4.0	+4.7	1.5
Siltstone (Roof)					
		82-4	+3.1	+3.3	1.2
Mudstone (Floor)					
15	3W-40		-0.00	-0.19	6.3
18	1S-76		-0.33	-0.48	8.0
19	2S-27		-0.05	-0.25	10.8
24	1S-76		-0.00	-0.19	3.9
25	2S-27		-0.25	-0.40	11.6

TABLE 5-9

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM - KIPP MINE

TEST RESULTS - BULK DENSITY

SAMPLE NO.	LOCATION	DRILL HOLE #	DISTANCE ABOVE (+) OR BELOW (-) GALT (m) From	To	BULK DENSITY
Mudstone (Roof)					
		82-2	+2.6	+2.7	2.92
		82-4	+4.0	+4.7	2.39
Siltstone (Roof)					
		82-4	+3.1	+3.3	2.47
Sandstone (Roof)					
		82-2	+7.5	+7.6	2.51
Carbonaceous Mudstone (Floor)					
	2S-112		-0.00	-0.10	1.58
Mudstone (Floor)					
	3W-40		-0.00	-0.19	1.88
	1S-76		-0.33	-0.48	2.23
	2S-27		-0.25	-0.40	2.21
Siltstone (Floor)					
		82-3	-3.60	-3.75	2.46
		82-3	-9.20	-9.35	2.42

TABLE 5-10
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM - KIPP MINE

TEST RESULTS X-RAY DIFFRACTION AND PETROGRAPHIC ANALYSES

Sample #	Distance (m) Above(+) or Below(-) Galt Seam	Quartz	Plagio-clase	Carbon-aceous Fragments	Illite/ Muscovite	Kaolinite	Montmoril-lonite	Calcite
SA 1	+0.35	-(3%)[1]	-(3%)[1]	3%[1]	-	-	-	-
SA 2	+0.25	+(25%)[1]	+(25%)[1]	10%[1]	-(2%)[1]	-	+	
SA 5	Parting**			3%			++	
SA 18	-0.33 to -0.45	(3%)[1]	(3%)[1]	20%[1]	-	-	++	
SA 19	-0.05 to -0.25	+(1%)[1]	-(1%)[1]	20%	-	-	++	
SA 38	-0.05 to -0.27	(2%)		30%[1]	-(½%)[1]	-	++	
SA 39	-0.27 to -0.35	(5%)[1]	(5%)[1]	25%[1]	-		++	
SA 77	+3.0 (824)	+	-		-	-	+	
SA 78	+6.2 (82-4)	+	-		-	-	+	
SA 79	+11.2 (82-4)	+	tr		-	-	+	

++ = dominant

+ = major

- = minor

* = not x-rayed for

[1] Visually estimated mode based on framework only. Percentage shown for quartz or plagioclase is the combined quartz and plagioclase content of the framework grains only. Percentage shown is for framework muscovite only.

** 1.3 m above base of Galt

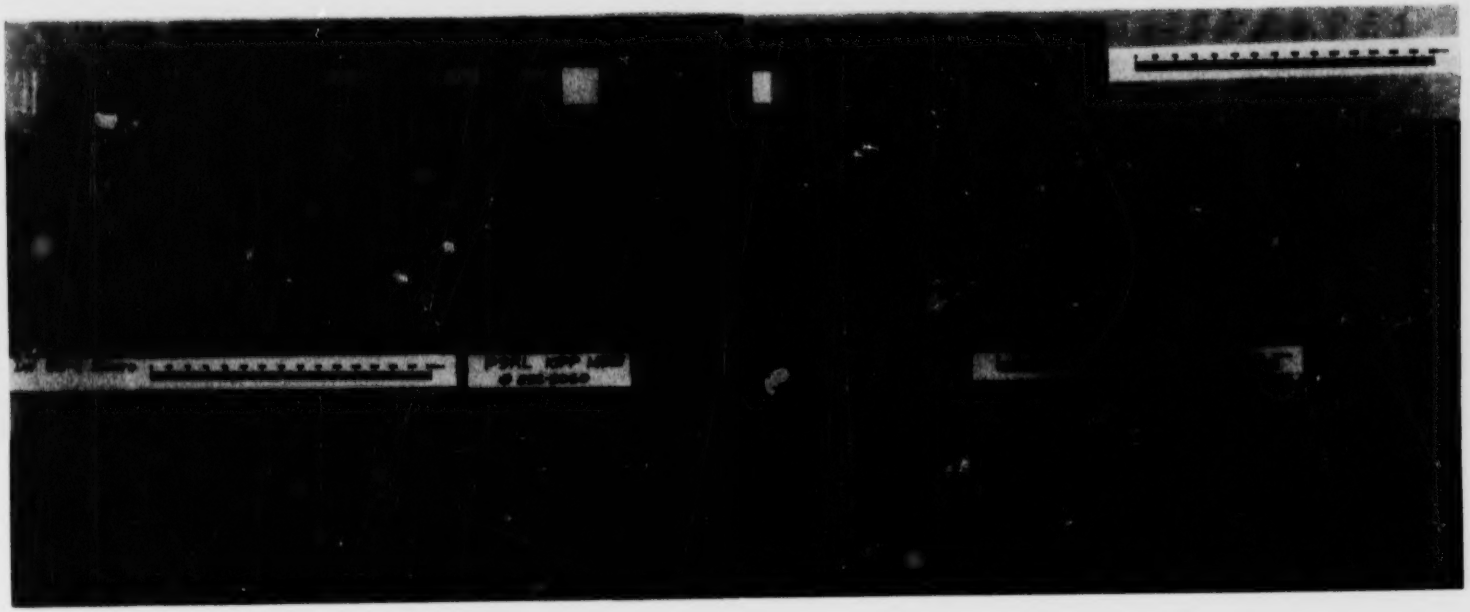


FIGURE 5-1
BH 82-1

Borehole 82-1

Location 5 514 954 mN, 74 895 mE

Direction -90°

FIGURE 5-1
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM – KIPP
CORE PHOTOGRAPH BH 82-1

PA73-24A

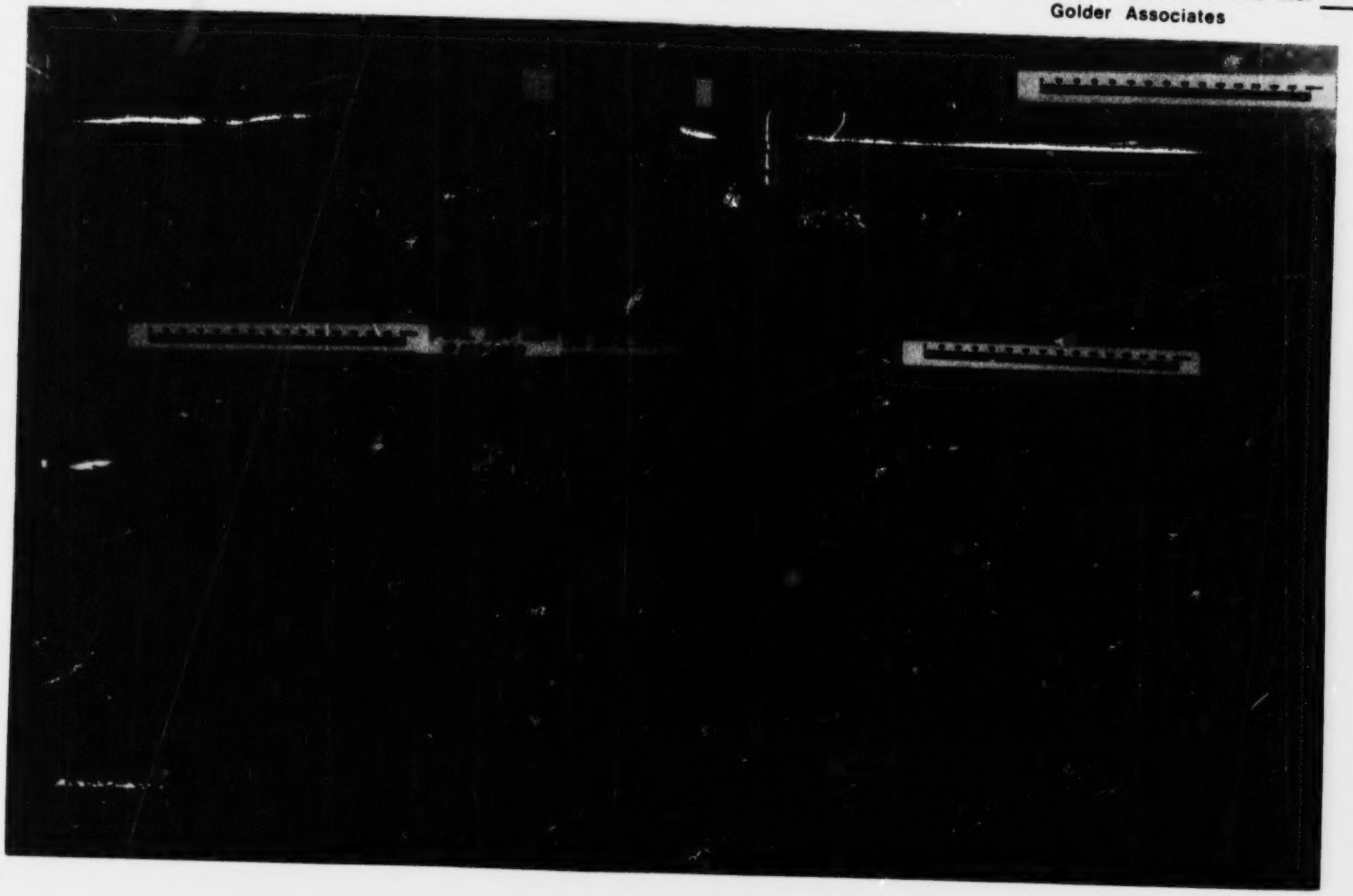


FIGURE 5-2
BH 82-2

Borehole 82-2

Location 5 514 954 mN, 74 895 mE

Direction + 90°

FIGURE 5 - 2
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM – KIPP
CORE PHOTOGRAPH BH 82-2

PA73-25A

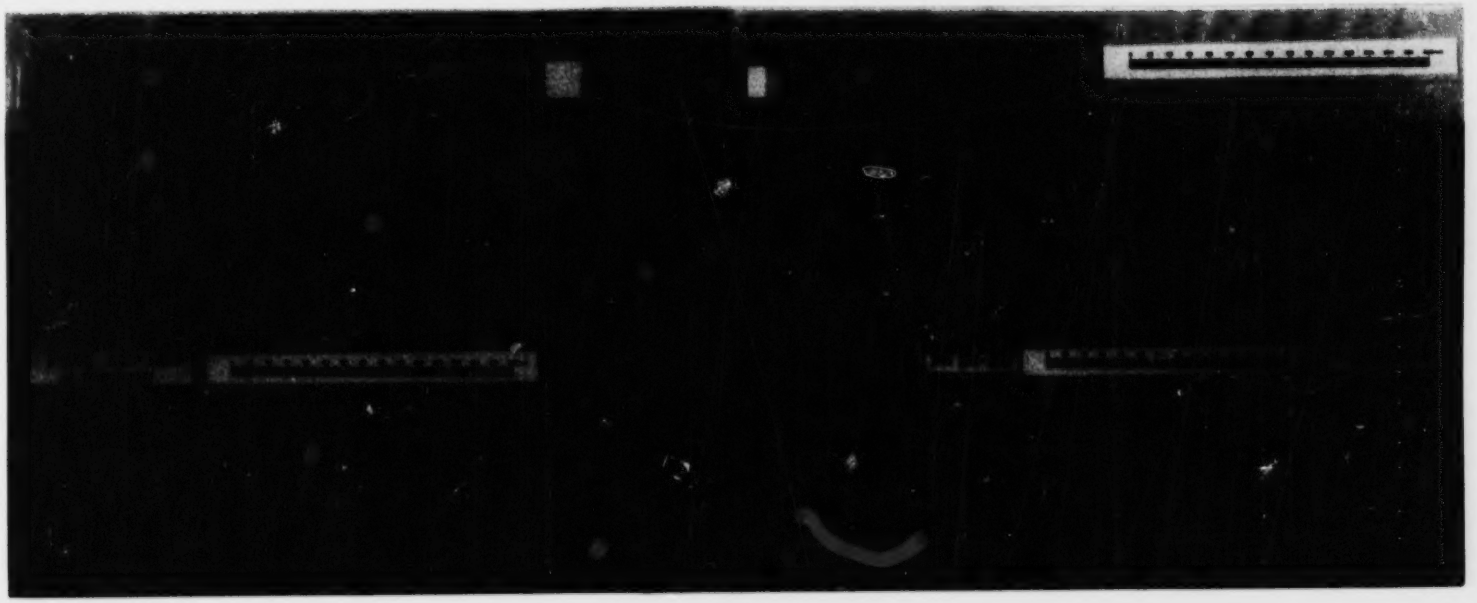


FIGURE 5-3
BH 82-3

Borehole 82-3

Location 5 514 829 mN, 74 887 mE

Direction -90°

FIGURE 5-3
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM – KIPP
CORE PHOTOGRAPH BH 82-3

PA73-26A

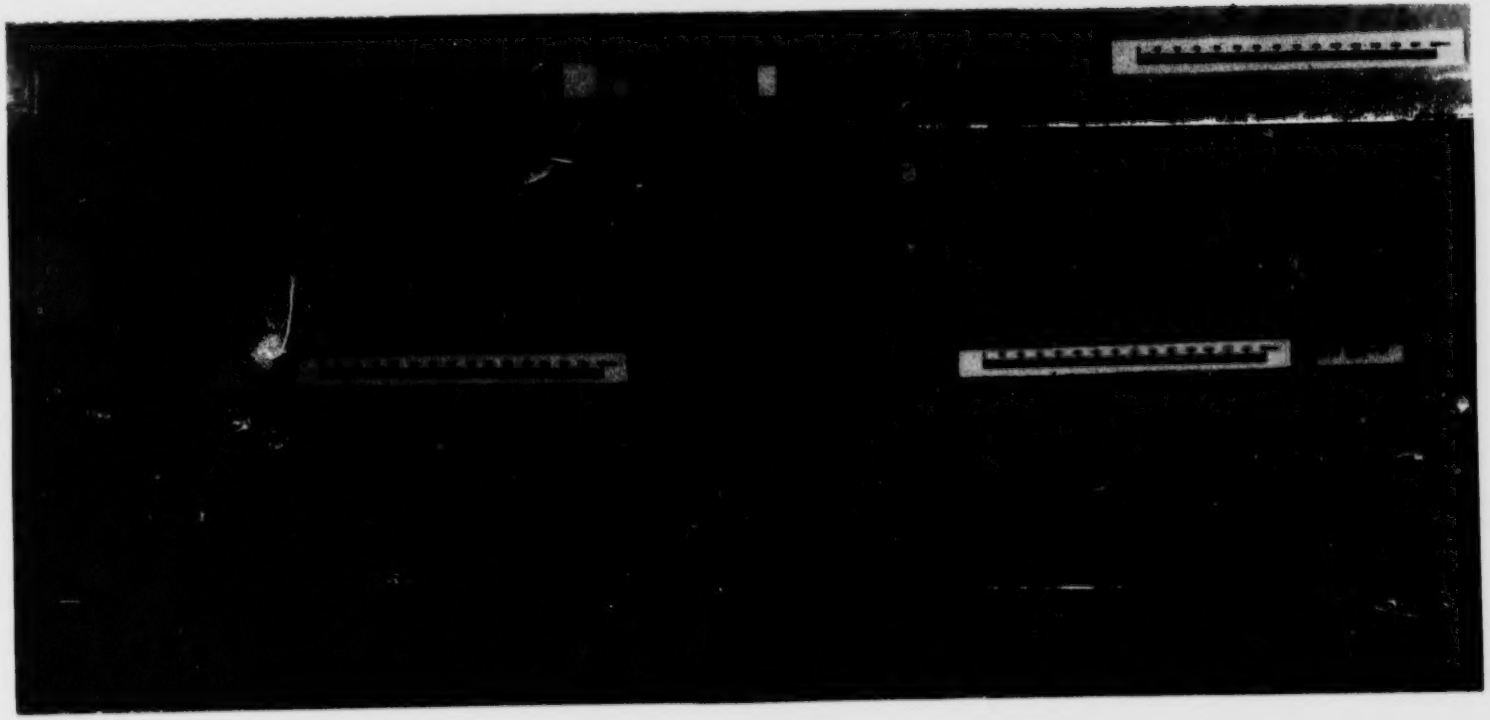


FIGURE 5 - 4
BH 82 - 4

Borehole 82-4

Location 5 514 829 mN, 74 887 mE

Direction +90°

FIGURE 5 - 4
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT
TRIAL MINING PROGRAM – KIPP
CORE PHOTOGRAPH BH 82-4

PA73-27A

Depth m	Core Run No.	Recov. %	Lithologic Description	Log	Structure: Gouge (mm) Broken Zones	Structure: Bdg. Brk. Int. cm	Structure: Fractures: Roughness	Structure: Fractures: Dip	Structure: Fractures: Notes	Rock Quality Designation % (25 50 75)	Test Results & Notes: Depth No.	Comp. Strength MPa	Slake Durability Elastic Modulus	Rock Class
20			END OF BOREHOLE											
19	16	100	Hard, medium grey MUDSTONE; massive (Bearpaw Formation)							100				
18	15	100	soft BENTONITE							100				
17														
16	14	100								100				
15														
14	13	100				50 to 100				100				
13	12	90	As above				/R	0°		90				
12	11	100								100				
11						>100								
10	10	100	Hard dark grey, fine silty SILTSTONE (Oldman Formation) becoming less silty with depth							100				
9	9	100				>100	/R	5°		100				
8			Hard, light greenish grey SANDSTONE, wavy bedded with thin silty laminae.			50 to 100				100			UC. = 35.5 MPa B.D. = 2.51	
7	8	100	MAIN RIDER hard black COAL SEAM; Carb. MUDSTONE			5				0				
6	7	100	Medium hard, medium grey SILTSTONE; finely laminated, arenaceous			50 to 100				100				
5			Medium hard, dark brown grey, carbon bearing MUDSTONE; massive; CARB. SHALE; Medium hard dark grey MUDSTONE; massive, lenticular		XXXX					0 / 40 / 100				
4	6	120	laminated; numerous arenaceous laminae.			50				100			UC. = 53.0 MPa F.S. = 4.8%	
			arenaceous			30				100				

			arenaceous			50 to 100				100				
5	7	100	Medium hard, dark brown grey, carbon bearing MUDSTONE; massive		XXXX XXXX XXXX XXXX									
			CARB. SHALE											
			Medium hard dark grey MUDSTONE; massive											
			lenticular							100				
4	6	120	laminated; numerous arenaceous laminae.			50				100				U.C. = 53.0 MPa F.S. = 4.8%
3			Hard, dark brown grey MUDSTONE; massive		XXXX XXXX									
			COAL			10 to 30								
	5	100	dark grey with arenaceous laminae							100				U.C. = 55.9 MPa B.D. = 2.92
			hematitic											
2														
	4	100	dark grey							70				
	3	100	medium grey, laminated.			30 to 50				70				
1	2	80	occ. arenaceous laminae							80				
	1	55	carbon bearing, massive			20								
			Med. hard, dark GALT SEAM COAL brown to black CARBONACEOUS SHALE with thin coal laminae.			5				25				
0			Top of GALT SEAM											
			B.H. 82-2 / B.H. 82-1											
0			Base of GALT SEAM											
			Med. hard, med. grey silty MUDSTONE with arenaceous laminae											
1	1	75	carbon bearing			5 to 10				0				S.D. = 9.62%, 92.2%
	2	55	hard, carbonaceous with coal laminae							15				
			Hard, black COAL											
	3	100	Med. hard, greyish brown CARB. MUDSTONE			4								
			BENTONITE				1s	5°		55				
2	4	120	Med. hard, med. grey, arenaceous MUDSTONE, bent. bearing; thin coal parting			10				75				
	5	95	Hard, light grey, silty SANDSTONE; bent. bearing, wavy bedded to laminated							85				U.C. = 54.4 MPa
3	6	100	numerous hematitic bands							100				
			numerous coal laminae											
4			Hard, greenish grey, arenaceous SILTSTONE, numerous coal laminae.											
	7	95	medium hard, grey MUDSTONE							75				
5			wavy bedded, light grey											
			less sandy			10								
6	8	90				10 to 20	1s	55°		80				
			Medium hard, carbon bearing MUDSTONE with occ. carb. mudstone partings; friable											
	9	100	Med. hard, med. grey SILTSTONE; friable, brecciated.		XXXX XXXX XXXX XXXX XXXX XXXX XXXX XXXX		1k	45°		100				
7			dark grey							0				
8	10	90	Medium hard, medium grey MUDSTONE			30 to 50				100				
			Hard, light grey, silty SANDSTONE wavy bedded.											
9														
	11	100	slump features							100				
10			Hard, light grey massive SANDSTONE											
			END OF BOREHOLE											

ABBR
BD
UC
FR
SD

NOTE
1. Se

FRAC

LEGE
m3

DRILL
Drill
Bit -
Cont
Date
Date
Loca
Drillh
Core

DRILLING AND SURVEY DATA

Drill Rig - Diamond Winkie
Bit - Inserted Diamond.
Contractor - Drillcor, Langley, B.C.
Date Drilled - Sept. 7 - 10, 1982 by G. Doubt
Date Logged - Sept. 7 - 10, 1982 by W. Claevs
Location - Kipp Mine, 2 South Road - 5 514 954 mN, 74 895 mE
Drillhole Orientation - Vertical
Core Barrel - Double Tube, NX.

LEGEND

⊠ Core loss - assumed to have occured at top of run.
m3 Gouge with thickness in millimeters, t = trace.
xxx Broken or Brecciated
HHH Broken - mainly horizontally
CIB Concretionary Ironstone band

FRACTURE ROUGHNESS

Macro		Micro	
⌒	Planar	K	Slick
⌣	Curve	S	Smooth
⌇	Stepped	R	Rough
≈	Irregular		

NOTE

1. See also core photographs in FIGURES 5-1 and 5-2

ABREVIATIONS

BD Bulk Density
UC. Uniaxial Compressive Strength
F.R. Free Swell
S.D Slake Durability

FIGURE 5-5

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM — KIPP

GEOTECHNICAL CORE LOG
BOREHOLES 82-1, 82-2

PA73-028A

Depth m	Core Run No.	Recov. %	Lithologic Description	Log	Structure: Gouge (mm) Broken Zones	Structure: Bdg. Brk. Int. cm	Structure: Fractures: Roughness	Structure: Fractures: Dip	Structure: Fractures: Notes	Rock Quality Designation %	Test Results & Notes: Depth No.	Test Results & Notes: Comp. Strength MPa	Test Results & Notes: Slake Durability Elastic Modulus	Test Results & Notes: Rock Class
			END OF BOREHOLE (equipment breakdown)							25 50 75				
14			Medium hard, dark grey slightly laminated. MUDSTONE. (Bearpaw Shale)											
	12	90				>100				90				
13														
12	11	100	massive, dark grey MUDSTONE			>100				100				
11														
	10	100	Hard, medium grey, massive SILTSTONE							100				
10			arenaceous											
9	9	105	light grey, wavy bedded, grading downward to massive silty SANDSTONE							105				
			Hard, medium grey, wavy bedded, arenaceous SILTSTONE.											
8						>50				100				
	8	100	MAIN RIDER SEAM: COAL; arg. COAL; carb. MUDSTONE; COAL			5				0				
7			Medium hard, medium grey MUDSTONE, slightly laminated.			3				30				
						10				100				
						50 to 100								
6					30									
	7	95	Medium hard dark grey arg. COAL		10					60				
			carbon bearing MUDSTONE		20									
			Medium hard, dark grey MUDSTONE, massive		5 to 10									
5			concretionary ironstone laminated		XXXX					0				
					XXXX								B.D. = 2.39	
					XXXX								F.S. = 1.5%	
	6	100			XXXX	10 to 20				80				
4			concretionary ironstone, v. hard		XXXX									
					XXXX									
			COAL											
3			Medium hard, medium grey SILTSTONE, finely laminated.			10 to 20				100			U.C = 82.1 MPa	
	5	85											B.D. = .47	
			Medium hard, dark grey MUDSTONE			20							F.S. = 1.2%	
2														
			COAL			5								
			carbonaceous		XXXX									
	4	100			XXXX					0				
	3	100			XXXX									
1			slightly laminated		XXXX									
	2	30			XXXX									
	1	100			XXXX									
			GALT RIDER Hard black COAL		XXXX									
0			Top of GALT SEAM											
			B.H. 82-4 / B.H. 82-3											
			Base of GALT SEAM											
0														
	1	100	Medium hard, dark grey massive MUDSTONE		10	10	1S	60°		0				
1	2	60	Medium hard, medium grey, arenaceous SILTSTONE;			10	1S	45°		35				

UC. = 41.1 MPa
B.D = 2.46

UC. = 42.0 MPa
B.D = 2.42

| | | |

FIGURE 5-7

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

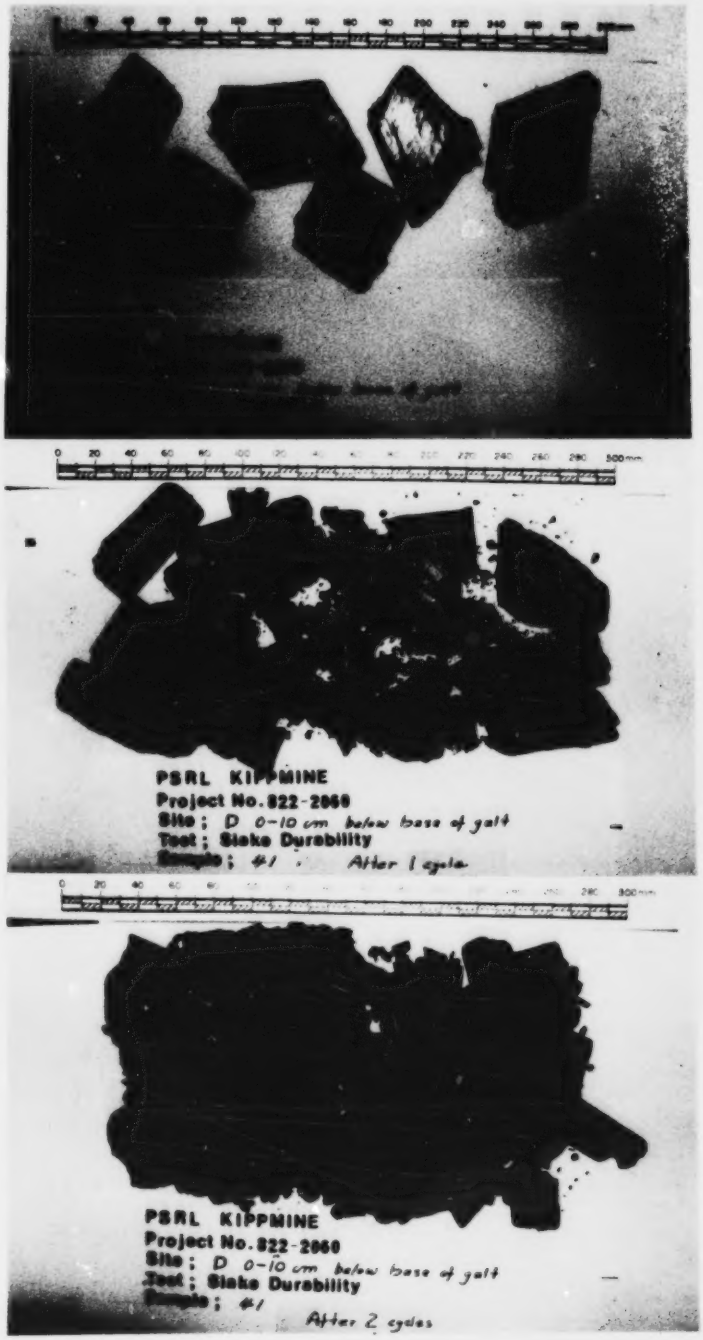
PICTURE BUTTE COAL PROJECT

S.D. Index (2nd Cycle) 64 %

TRIAL MINING PROGRAM – KIPP

SLAKE DURABILITY TEST

COAL AT BASE OF GALT SEAM


PSRL KIPPMINE
Project No. 822-2060
Site ; D 0-10 cm below base of galt
Test ; Slake Durability
Sample ; #1 After 1 cycle
PSRL KIPPMINE
Project No. 822-2060
Site ; D 0-10 cm below base of galt
Test ; Slake Durability
Sample ; #1
After 2 cycles

S.D. Index (2nd Cycle) 56 %

FIGURE 5-8

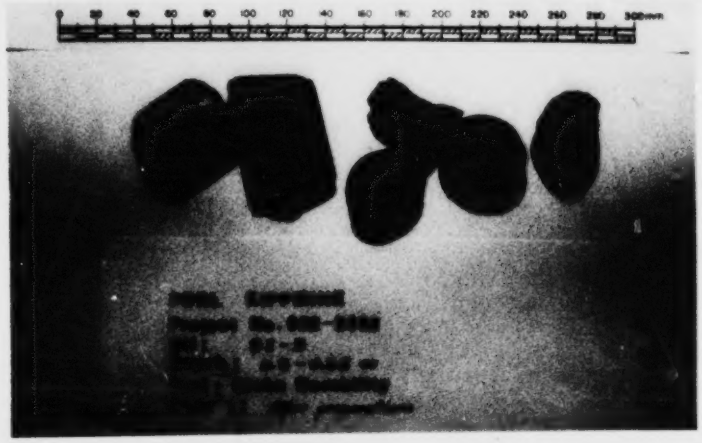
PRECAMBRIAN SHIELD RESOURCES LIMITED

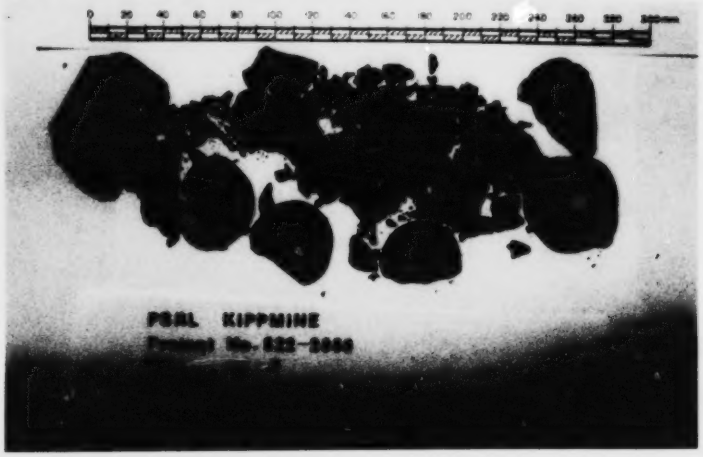
COAL DIVISION

PICTURE BUTTE COAL PROJECT

TRIAL MINING PROGRAM – KIPP

SLAKE DURABILITY TEST
CARBONACEOUS MUDSTONE
0.0-0.10m below GALT SEAM





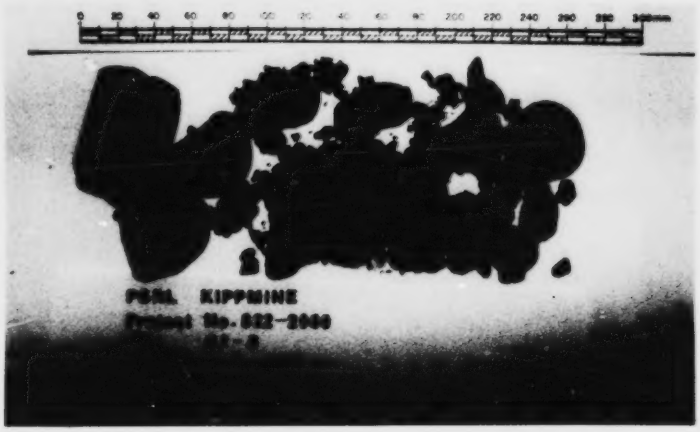


FIGURE 5-9

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

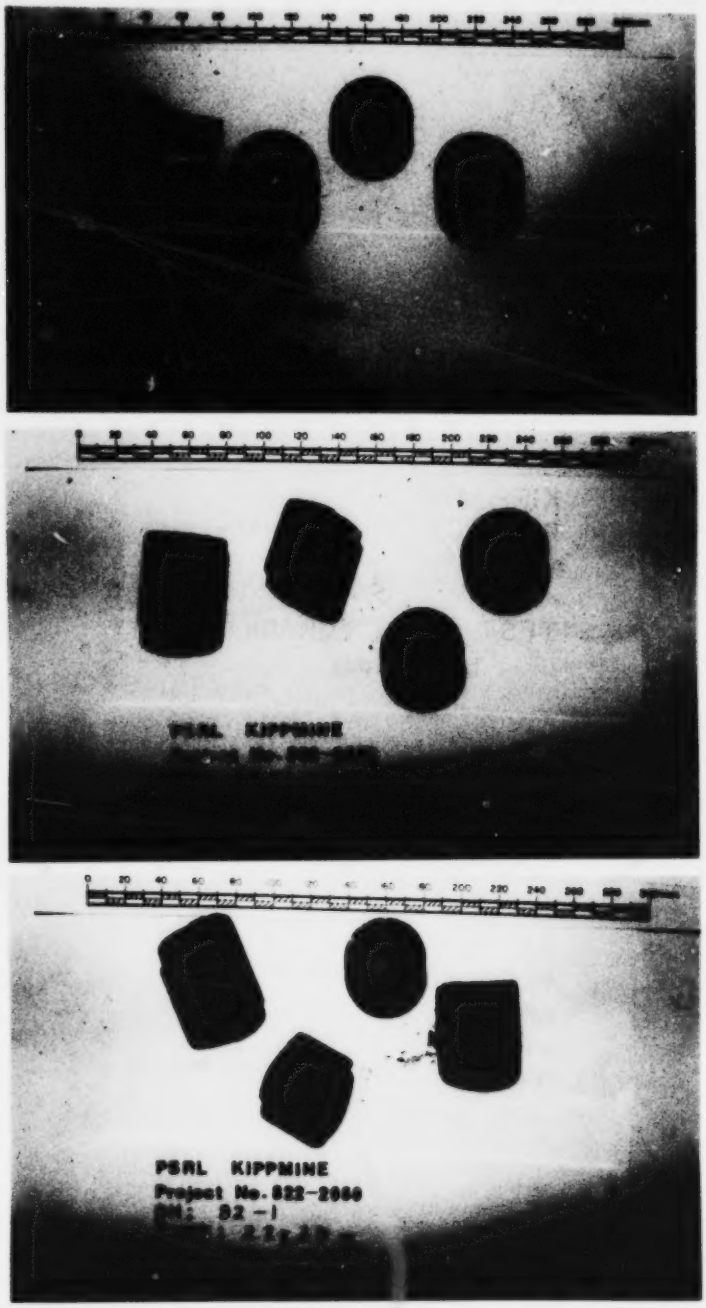
S.D Index (2nd Cycle) 92 %

TRIAL MINING PROGRAM – KIPP

SLAKE DURABILITY TEST

SILTY MUDSTONE

0.8 - 0.85m below GALT SEAM



S.D. Index (2nd Cycle) 93 %

FIGURE 5-10

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

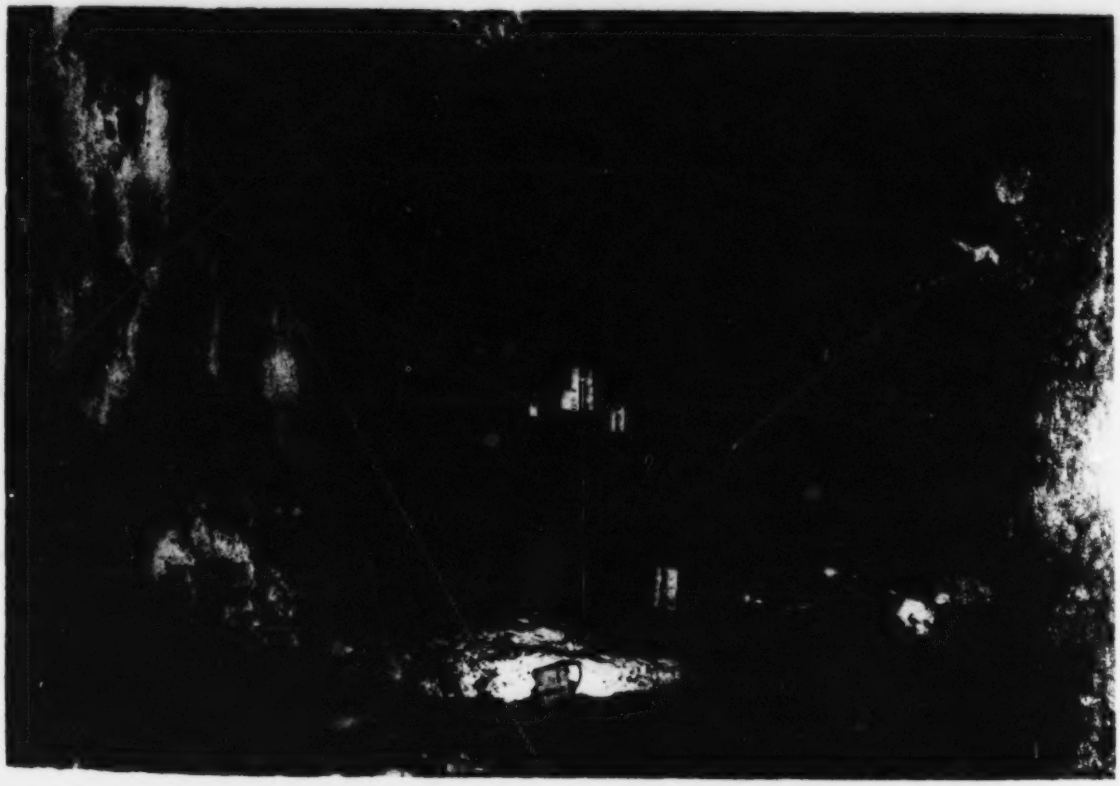
TRIAL MINING PROGRAM – KIPP

SLAKE DURABILITY TEST

SILTY SANDSTONE

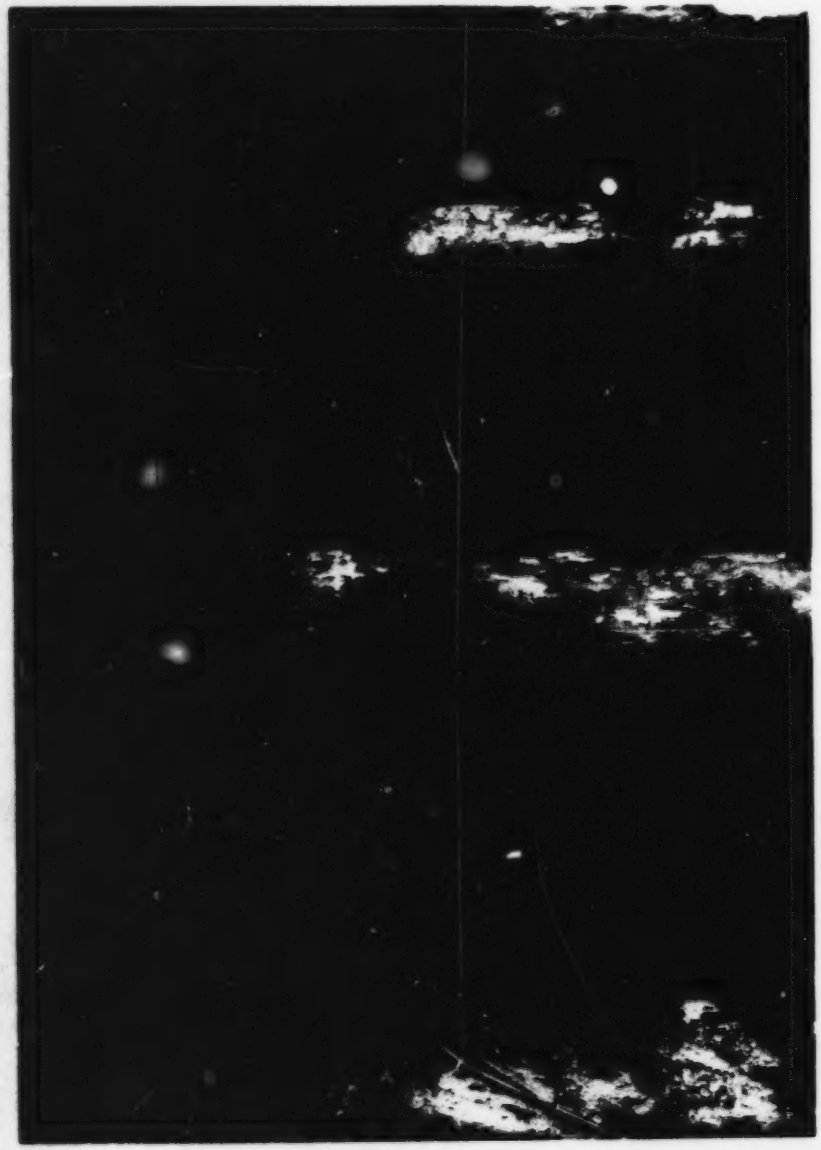
2.2 - 2.3m below GALT SEAM

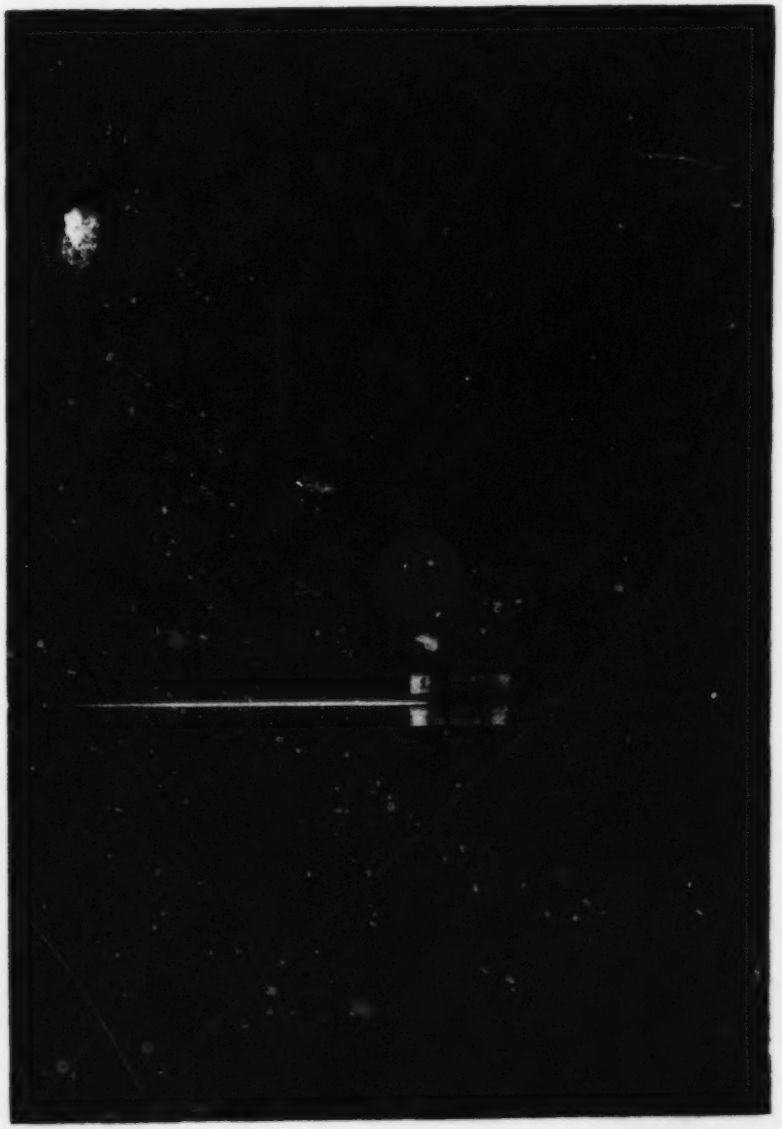
APPENDIX A - PHOTOGRAPHS

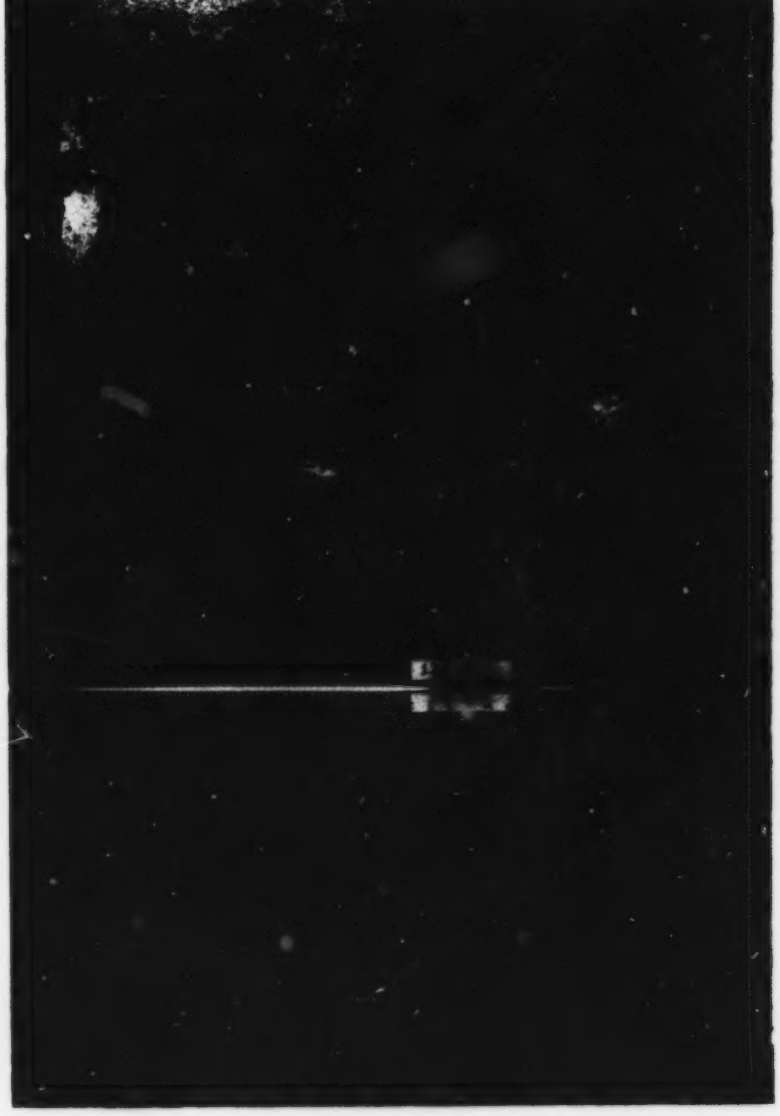


I SOUTH ROAD TEST SECTION
Looking south along east rib Nov. 5, 1982; one month after steel girders had been removed.

I SOUTH ROAD TEST SECTION
As above Dec. 7, 1982; two weeks after the roof fall.







I SOUTH ROAD TEST SECTION
Looking south along west rib Nov. 5, 1982; one month after steel girders had been removed.

I SOUTH TEST SECTION
As above Dec. 7, 1982, two weeks after the roof fall.

Number/
Numéro

2

A GEOLOGICAL EVALUATION OF THE TECHNICAL FEASIBILITY OF LONGWALL MINING IN THE GALT SEAM AT KIPP TRIAL MINE NUMBER 1787/E/01 (UP-P-137)

INTERIM REPORT

BY

PRECAMBRIAN SHIELD RESOURCES LIMITED

Work on this project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada.

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Set 16

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT
OF
PICTURE BUTTE

PREPARED FOR

PRECAMBRIAN SHIELD RESOURCES LIMITED

SUBMITTED BY

ASSOCIATED MINING CONSULTANTS LTD.

GOLDER ASSOCIATES

E ASSOCIATION OF
LESSIONAL ENGINEERS,
CISTS and GEOPHYSICISTS
OF ALBERTA
ERMIT NUMBER
P 2361
ssociated Mining
Consultants Ltd.

THE ASSOCIATION OF
PROFESSIONAL ENGINEERS,
GEOLOGISTS and GEOPHYSICISTS
OF ALBERTA
PERMIT NUMBER
P 949
GOLDER ASSOCIATES
(WESTERN CANADA) LTD.

JANUARY 1983

AR-61B

Associated Mining Consultants Ltd.
Golder Associates

P.O. Box 6500, Station D, 840 - 7 Avenue S.W., Calgary, Alberta T2P 2V8
Tel: (403)264-9496 Telex: 038-22761

January 31, 1983

FILE: 0302-PA73
0201-1607

REF: AR-61B

Precambrian Shield Resources Limited
3rd Floor Edinburgh Place
900 - 6 Avenue SW
CALGARY, Alberta
T2P 3K2

Attention: Mr. J.H.H. Chamberlin, P.Eng.
Vice President, Coal

Dear Sirs:

Re: Picture Butte Coal Project
A Technical Assessment of Picture Butte

We are pleased to submit our preliminary report **'Picture Butte Coal Project - A Technical Assessment of Picture Butte'.** This study has enabled us to evaluate mining potential at Picture Butte.

Extrapolating the information gained at Kipp to Picture Butte we conclude that:

- Inseam physical characteristics and the behaviour of the coal and proximate roof and floor strata at Kipp indicate that with proper selection and application of today's coal mining technology, a longwall system of mining is possible at Picture Butte.

- The geologic correlation indicates that the roof and floor strata at Picture Butte will be similar to that encountered at Kipp, although preliminary indications from geotechnical data correlation are that ground conditions encountered at Kipp are better than those anticipated at Picture Butte.

.....Page 2

However, it is significant that geotechnical data from Picture Butte was acquired over a very limited area and, therefore, may not be fully representative of ground conditions over the entire Picture Butte property. Further geotechnical studies are to be carried out as part of this program and should provide a better evaluation of general conditions over the property. It should be noted that actual ground conditions encountered at Kipp were better than those anticipated from geotechnical data collected prior to shaft sinking and mining.

- The existence of any buried channels or major aquifers lying above the Galt Seam across the Picture Butte property is considered extremely unlikely.

- The anticipated low permeability of the overlying bedrock and glacial overburden should minimize any ground water inflow from the overburden into mine workings. Further, it is reasonable to assume that hydrogeologic conditions within the Galt Seam at Picture Butte will be the same as at Kipp. Large quantities of groundwater flow through the Galt Seam at Picture Butte are, therefore, not anticipated.

- Alternate methods of mining should be considered. It is possible that when all relevant parameters to mine design are investigated, some areas of the Picture Butte property will not be amenable to mining by the longwall system.

- The preferred direction of main entry driveage is north to south or south to north. Main entries supported by arches should be driven as small as practicable, and support should be complemented by roof bolts where required. Special types of intersection support systems can be designed for the sizes of entries required.

- Service roadways for longwall retreat should be driven in an east to west or west to east direction, as narrow as possible, and rectangular in shape.

.....Page 3

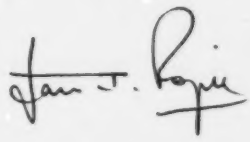
We hope this report meets your immediate requirements. If you have any questions regarding the contents of the report please do not hesitate to contact us.

Respectfully submitted,

ASSOCIATED MINING CONSULTANTS LTD.

Hans U. Bielenstein, P.Geol.
Senior Geologist

GOLDER ASSOCIATES



Ian T. Rozier, P.Eng.
Senior Engineer

HUB/ITR:ms

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

Table of Contents

Letter of Transmittal

Summary of Findings, Conclusions and Recommendations

SECTION 1 Introduction

SECTION 2 Description of the Picture Butte Project Area

SECTION 3 Results of Trial Mining Program

SECTION 4 Correlation of Data from Kipp to Picture Butte

SECTION 5 Underground Mining at Picture Butte

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SUMMARY OF FINDINGS, CONCLUSIONS AND RECOMMENDATIONS

Geotechnical data in the Picture Butte area is limited, and is confined to a relatively small area. In order to formulate preliminary mine design criteria it is necessary to extrapolate this data over the entire Picture Butte area.

The observations and data obtained from the TMP at Kipp Mine, together with correlation of geotechnical data from that area and the limited geotechnical data from the Picture Butte area indicate the following:

a. Inseam physical characteristics and behaviour of the coal and proximate roof and floor strata at Kipp indicate that with proper selection and application of today's coal mining technology, **a longwall system of mining is possible at Picture Butte.**

b. Main entries supported by arches should be driven as small as practicable and support should be complemented with roof bolts where required. The preferred direction of main entry driveage is south to north or north to south. Special types of intersection support systems can be designed for the sizes of entries required.

c. **Service roadways for longwall retreat should be driven in an east to west or west to east direction,** as narrow as possible, and rectangular in shape. Roof bolts adjacent to the ribs of the retreat pillar of coal to be mined should be angled over the rib to help prevent roof fractures parallelling the rib line at or near the face, thereby reducing dilution of the coal.

d. For the longwall advance system service roadways must be of the arch type, supported by rigid or yielding arches of the correct support capacity. Roadside packs of hardwood, anhydrite or aqua-packs can provide adequate roadway stability. If legislation permits, shearers capable of cutting roadway arch profiles could be used to drive these service roadways in line with the advance of the longwall face.

e. Alternate methods of mining should be investigated. When all relevant parameters to mine design are explored, it is possible that some areas of the Picture Butte property will not be capable of being mined using the longwall system.

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SECTION 1 - INTRODUCTION

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1.2	References	1-2

FIGURES
(Following Text)

1-1 Location Map

SECTION 1 - INTRODUCTION

1.1 INTRODUCTION

Associated Mining Consultants Ltd. with Golder Associates were retained by Precambrian Shield Resources Limited (PSRL) to undertake a Geotechnical/Strata Control Trial Mining Program in the Galt Seam at Petro-Canada's Kipp Mine and to provide preliminary mine design parameters for mining of the Galt Seam at Picture Butte (Figure 1-1).

The terms of reference for this study are outlined in a joint proposal submitted to Precambrian Shield Resources Ltd. by AMCL and Golder Associates, dated March 1982, as amended in April, 1982 and again in September 1982. The basic objectives of the program were as follows:

- Evaluate available data from the Lethbridge area.
- Implement a geotechnical data collection program at Kipp Mine.
- Carry out a trial mining and monitoring program at Kipp to test relevant geotechnical/strata control features.
- Evaluate the results of these studies and formulate preliminary mine design parameters applicable to mining at Picture Butte.

The Galt Seam has previously been exploited in the Lethbridge area, the last mine closing in 1964. A literature study was carried out to document recorded mining conditions in this coalfield for comparison with those anticipated by previous geotechnical studies by Dames and Moore and Golder Associates in the Lethbridge area. The evaluation of previously accumulated data assisted in determination of geotechnical/strata control parameters at Kipp. The intent of the TMP was to

evaluate the potential for, and to determine appropriate mine design parameters applicable to the longwall advance and longwall retreat method of mining at Picture Butte.

The geotechnical/strata control project incorporated underground studies at the Kipp Mine and geologic/geotechnical investigations at Picture Butte. The project schedule was such that the studies at Kipp would take place during 1982, and the studies at Picture Butte during early 1983.

The first phase of the geotechnical/strata control project, the Trial Mining Program at Kipp, was completed on schedule in late 1982, and a report prepared entitled **'Picture Butte Coal Project - Trial Mining Program - Kipp'.**

This preliminary report presented here details the anticipated geotechnical/strata control conditions associated with mining of the Galt Seam at Picture Butte and will be amended when geotechnical data collection and evaluation is completed at Picture Butte. A final report will then be issued.

1.2 REFERENCES

The following references are used throughout this report.

1953 - 1971 F. Grant, et al. - (T1)
Untitled: Seven papers on roofbolting, subsidence, ground stress in Lethbridge and Canadian mines.

1947 I. Crawford
(1 Report) Thesis - University of Alberta.

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Evaluation of the Shaughnessy/Picture Butte Coal Properties - 1 Report, brown cover, 1 Legal File, green cover (backup).

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Geotechnical Review of Applicability of Shortwall Mining in the Lethbridge Coalfield.

1979 Dames & Moore (R14)
Report on Geotechnical Assessment for Shaft, Lethbridge Coalfield, Lethbridge, Alberta

1979A Norwest (R17)
1979 Data Package for the Lethbridge Coal Project.

1979B Norwest (R18)
Maps to Accompany 1979 Data Package for the Lethbridge Coal Project.

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Report on Subsidence Study, Lethbridge Coal Project, Lethbridge, Alberta.

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Review of Groundwater Hydrology Conditions, Lethbridge Coal Project, Lethbridge, Alberta.

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Untitled: Field Operations and Structural Geology - Site Report.

1980 Norwest (G. Jordan)
1980 Exploration Report on Lethbridge Property - 1 Report, 2 Folios Maps - Part I and II.

1980 Daryl Thomas (Petrofina)
Washability Analysis of Core, 1975 & 1980 - 1 Report (clear cover).

1981 Eric Sundgaard (Petrofina)
1981 Exploration Report & Appendices - 1 Report, 1 Folio - Maps.

1981 Daryl Thomas (Petrofina)
Laboratory Core Analysis and Washability Report - 1 Report (brown cover).

1981A Golder Associates
Report No. 8122028. To Petrofina Canada Inc. Geotechnical Evaluation of Bedrock, Proposed Decline and Test Mine, Lethbridge/Picture Butte Coal Property, Alberta.

1981B Golder Associates
Report No. 8122028A. To Petrofina Canada Inc. Geotechnical Investigation of Soil Overburden for Proposed Decline Tunnel at the Shaughnessy/Picture Butte Coal Property.

1982 Petro-Canada
Application to ERCB re TMP including Appendix I of letter to D. Henderson dated July 5, 1982 showing anticipated seam section and anticipated quality of ROM and copy of Drawing 6/2 showing support of Intersections.

1982 M.J. Hendry.
Hydraulic Conductivity of a Glacial Till in Alberta.

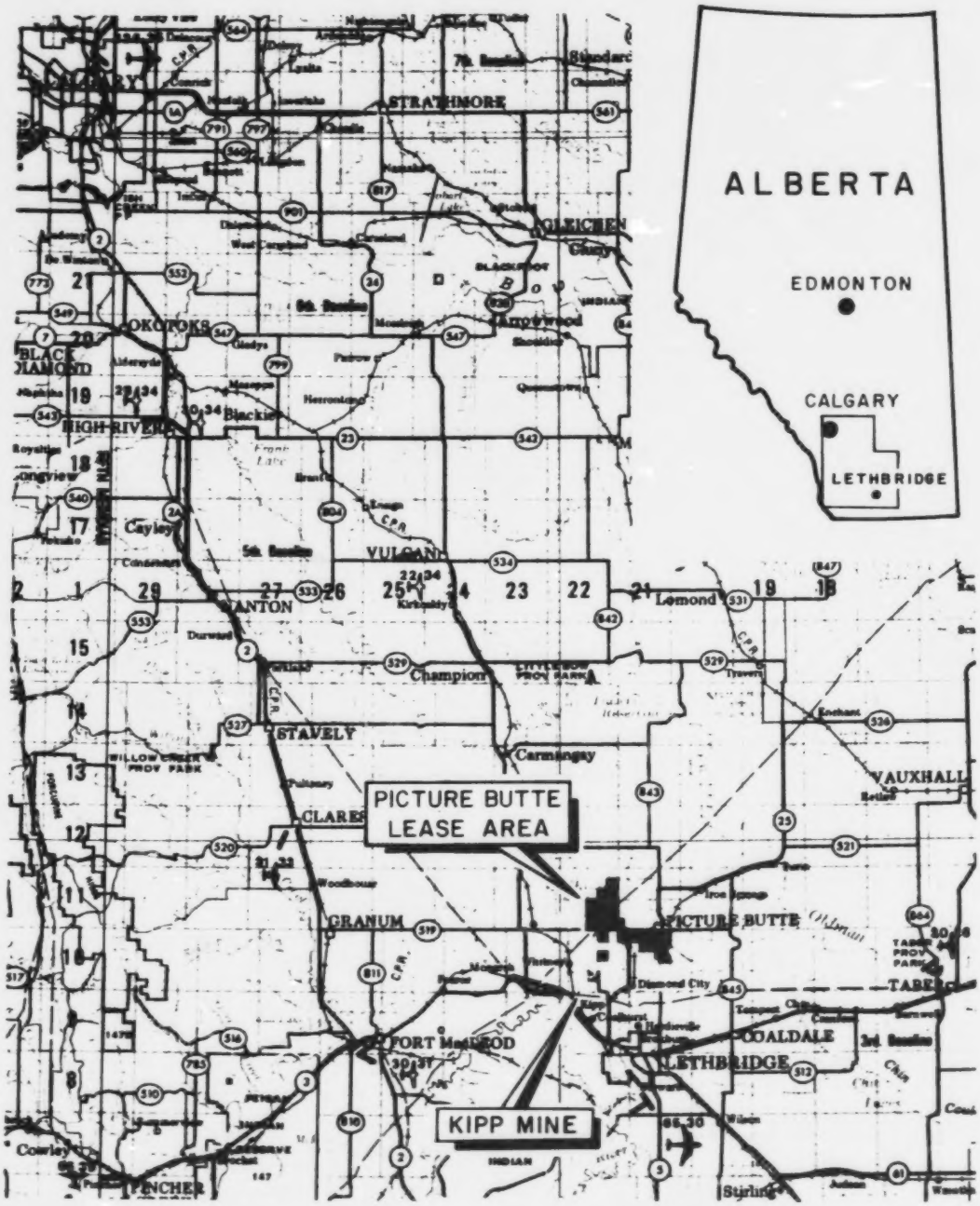

FIGURE 1-1

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

LOCATION MAP

PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SECTION 2 - DESCRIPTION OF THE PICTURE BUTTE PROJECT AREA

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2.4	Ground Water	2-2
2.5	Surface Water	2-3

FIGURES
(Following Text)

2-1 Picture Butte Property

2-2 Generalized Stratigraphic Section - Lethbridge Area

SECTION 2 - DESCRIPTION OF THE PICTURE BUTTE PROJECT AREA

2.1 LOCATION

The Picture Butte Property is situated to the west of Picture Butte, within the Lethbridge Coalfield some 20 km north of Lethbridge, Alberta. The extent of the PSRL lease is shown on Figure 2-1.

2.2 TOPOGRAPHY

The topography of the area consists of a gently undulating plain with a low, broad, northwest/southeast trending ridge to the northwest of the property which plunges towards the Picture Butte reservoir. Highway 519 cuts through the southern part of the property and several major irrigation canals flow through the area, supplying a network of smaller ditches which parallel section lines. Land use is primarily restricted to crop farming and, to a lesser degree, grazing.

2.3 GENERAL GEOLOGY

The Lethbridge Coalfield lies in the south-central Alberta plains, on the east flank of the Alberta Syncline; the regional dip is to the west. The bedrock in the general study area is overlain by a variety of glacial, fluvio-glacial and glacio-lacustrine deposits. The area immediately to the northeast, northwest and southeast of Picture Butte is directly underlain by glacio-lacustrine silts and clays which overlie a silty and clayey till deposited during the Wisconsin advance of the Keewatin continental ice sheet. The till material is locally derived, and therefore, its composition largely reflects that of the underlying strata, generally consisting of clay and silt, with frequent sand and gravel

layers. The glacial lake sediments are absent on the northwest-southeast trending ridge, and to the southwest of Picture Butte.

The bedrock in the area consists of massive mudstone of the Bearpaw Formation underlain by a sequence of laminated sandstone, siltstone, mudstone and coal of the Oldman Formation of the Belly River Upper Cretaceous Series. The upper part of the Oldman Formation contains numerous coal seams. The Galt Seam, which is the seam of economic interest, occurs near the top of the Oldman Formation. The general geologic sequence in the Lethbridge area is shown on Figure 2-2. The strata are generally flat lying, with a regional dip of at least 1 degree to the northwest.

A number of faults have been inferred from previous studies in the Lethbridge area, generally having a northwest to southeast strike direction. One major fault between Shaughnessy and the Kipp area has been interpreted to have a throw of approximately 30 m down on the south side. However, no major faults are thought to occur on the Picture Butte coal property. Mine plans from earlier operations indicate an almost random orientation for frequent faults with varying displacements, at the Standard Mine up to 3 m and at Galt #8 Mine up to 5 m.

2.4 GROUND WATER

The interpretation of the hydrology over the proposed mine area has limitations because of the sparse data. The conclusions drawn from previous studies in the Kipp and Picture Butte areas have been conflicting. Also, the anticipated hydrogeologic conditions of high flow through the Galt Seam predicted from previous surface exploration at Kipp prior to the sinking of the shaft were not encountered in the shaft sinking operations or in underground mining operations. Consequently, certain assumptions have been made relative to the effects of geology on the hydrogeologic regime over the area.

The detailed stratigraphy of the material overlying the bedrock in the Lethbridge area is unpredictable and difficult to correlate. Furthermore, buried channels that are known to exist beneath later deposits outside the immediate project area complicate the general hydrology of the Lethbridge area, although no such channels are thought to exist across the PSRL lease area.

2.5 SURFACE WATER

The Picture Butte area is traversed by an irrigation system consisting of ditches and canals. The hydraulic gradients of the flow system across the area are highly sensitive, with low water levels and nominal flows that supply large areas of farmland. Water levels and supply are controlled by a system of weirs and 'turn-out' water diversion structures that are constantly monitored to ensure adequate supply to farmland.

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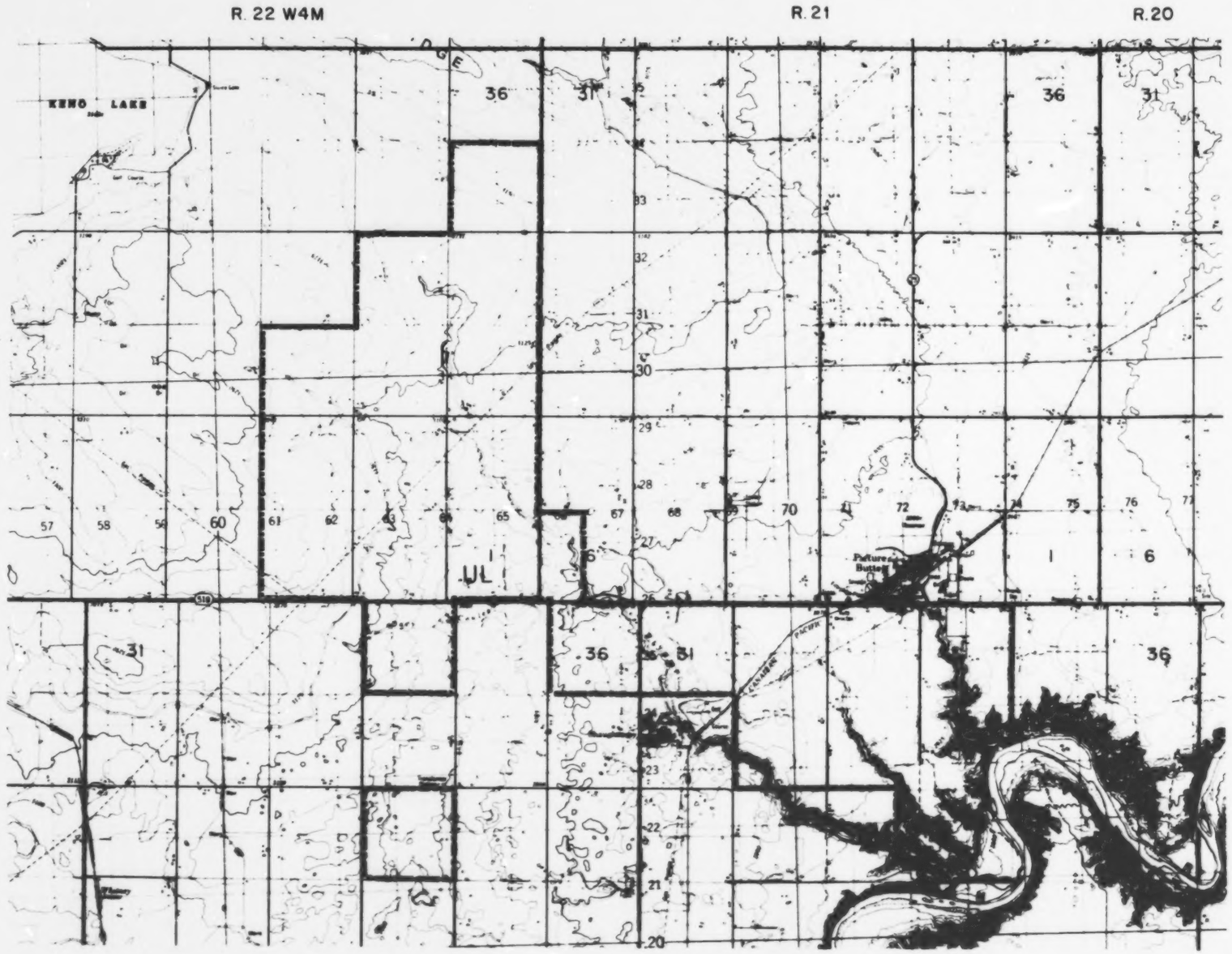

FIGURE 2-1

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

PICTURE BUTTE PROPERTY

PA73-888

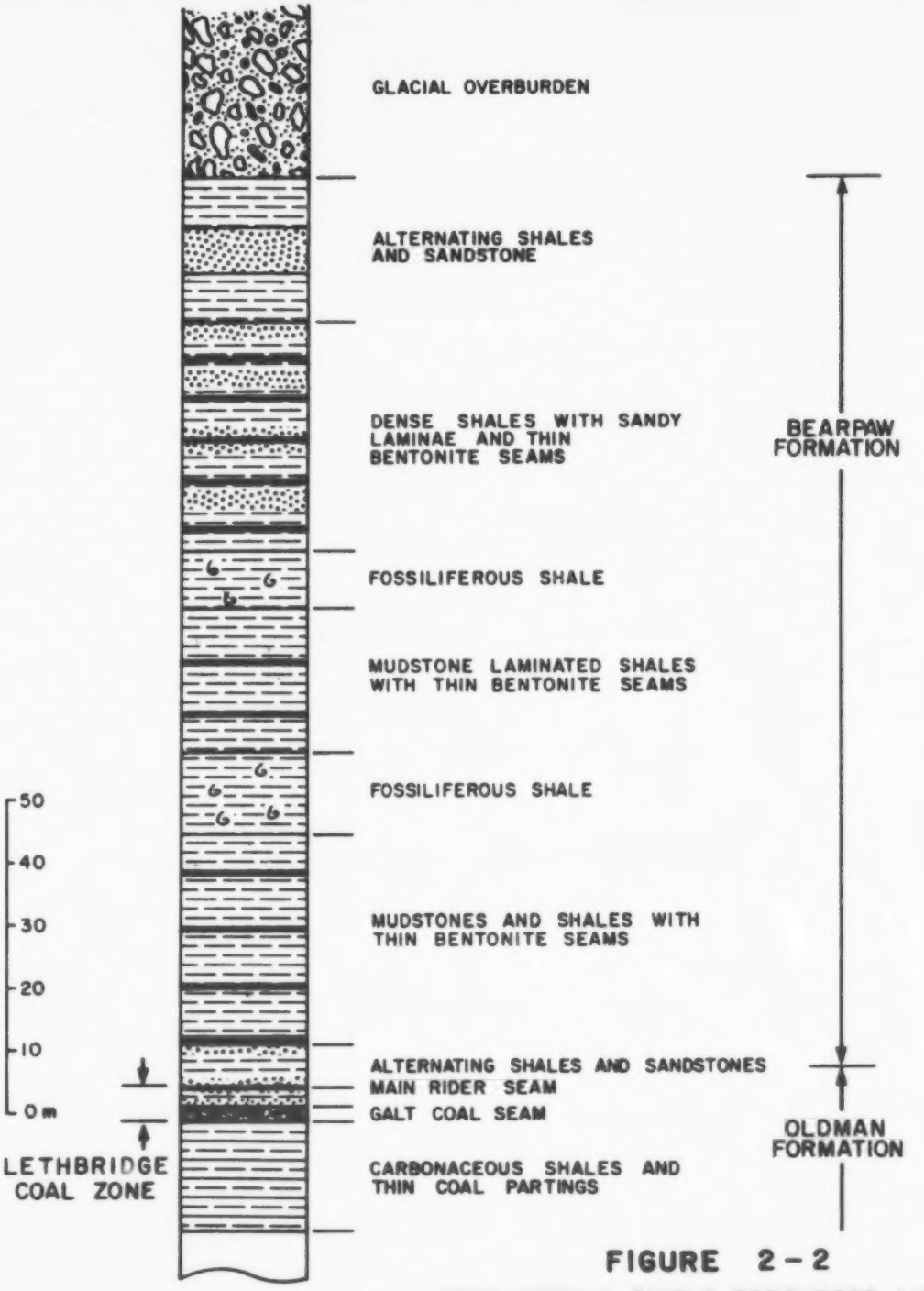

FIGURE 2-2

PRECAMBRIAN SHIELD RESOURCES LIMITED

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PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

GENERALIZED STRATIGRAPHIC SECTION LETHBRIDGE AREA

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SECTION 3 - RESULTS OF THE TRIAL MINING PROGRAM

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3.3	Roof Strata Behaviour	3-4
3.4	Conclusions	3-5

FIGURES
(Following Text)

3-1 Initial Roof Stability Plan
Trial Mining Program - Kipp

SECTION 3 - RESULTS OF THE TRIAL MINING PROGRAM

3.1 INTRODUCTION

The Trial Mining Program (TMP) at Kipp Mine added significantly to the existing data base for mining the Galt Seam. The actual TMP carried out from July to October 1982, is described in Report AR-61A **'Picture Butte Coal Project - Trial Mining Program - Kipp'.**

This report goes beyond the direct observations from the TMP to discuss some potential causes and effects of the strata behaviour observed at Kipp. Strata behaviour and its control is obviously site specific; however, there are some underlying behaviour characteristics that may be applicable to the mining conditions at Picture Butte.

This section gives our interpretation of some strata behaviour characteristics.

3.2 STRESS DISTRIBUTION

3.2.1 Mining Evidence

Roof conditions at the face of 1 South and 2 South roadways prior to the installation of the roof supports are shown on Figure 3-1. The immediate roof spalling, as the face was advanced, occurred in approximately one half of the total driveage of 1 South and 2 South and was confined principally to the east rib portion of the roadways. No immediate roof spalling was encountered in 3 West, 4 West and 5 West.

The TMP driveage started in I South, with a change in the mining horizon, at the old head end left by Petro-Canada in 1981 to 3 West. This portion of I South had immediate roof spalling conditions along the east rib. Initial roof support was roof bolts and steel straps, and steel girders. In the TMP steel straps were always used with roof bolts and subsequent mention of roof bolts should be taken to include steel straps.

3 West was then driven in a south-westerly direction parallel to the cleat with no immediate roof spalling. Initial support was roof bolts and steel girders.

2 South was then driven south past the future 4 West junction and even with a change in mining horizon no immediate roof spalling occurred as the roadway advanced. Initial support included roof bolts and steel girders.

I South was then driven south from 3 West to 4 West. The initial roof support over most of this driveage was roof bolts and wood booms. This support system was not adequate to prevent immediate roof spalling. The change over to steel girders from Set #62 to Set #76, 15 m of driveage, was unable to arrest the immediate roof spalling. All wood booms were replaced by steel girders and later centre-posted to secure this section of I South.

4 West was then driven in a westerly direction with no immediate roof spalling. Initial roof support was roof bolts and wooden booms.

2 South was then driven south from 4 West with a change in the mining horizon. Initial roof support was roof bolts and wood booms. Immediate roof spalling occurred as shown on Figure 3-1.

The wood booms had to be replaced by steel girders to secure this portion of 2 South roadway.

5 West was then driven east to 1 South with a change in the mining horizon. Initial roof support was roof bolts and wood booms. There was no immediate roof spalling during this driveage.

1 South was then driven north from 4 West to 3 West. Initial roof support was roof bolts and steel girders. The only immediate roof spalling occurred at the high stress breakthrough area at 4 West as shown on Figure 3-1.

5 West was then driven easterly from 1 South with another change in the mining horizon to completion of the TMP. Initial roof support was roof bolts and a combination of wood booms and steel girders. There was only a small amount of roof spalling in the vicinity of the fault as shown in Figure 3-1.

3.2.2 Deduced Stress Field

The immediate behaviour of the roof (very limited observation) tends to support a hypothesis of a stress field with the major axis plunging in a south-westward direction approximately parallel to the cleat and the minor axis approximately normal to the cleat. No quantitative figures can be assigned to the hypothetical stress field deduced from observations made in the TMP.

3.2.3 Discussion

Consider the directions of this deduced stress field (Figure 3-1) and its effect on the roof strata at the faces of the north-south, south-north driveages in the TMP.

a. I South/2 South - Advancing South

The high concentration of stress in the roof strata and coal face is just ahead of the east rib of the roadway; when the coal is removed this high stress in the roof strata causes failure of immediate roof strata, thereby initiating a roof failure mode along the east rib. The stress on the coal and roof strata just ahead of the west rib has been partially de-stressed a short distance into the coal face, therefore, when this coal is mined this roof remains stable. The roof failure mode initiated along the east rib, if artificial roof supports (roof bolts, booms, posts, etc.) are inadequate, will gradually lead to roof failure across the roadway.

b. I South Advancing North

The high concentration of stress in the roof strata and coal face is now just ahead of the west rib, somewhat reduced by the direction of the stress field which transmits stress from the roof strata to the west coal pillar, the roof along the west rib remains stable on mining advance. The stress on the coal face and roof strata just ahead of the east rib has been partially de-stressed a short distance into the coal face, therefore, when this coal is mined this roof remains stable. This section was supported by roof bolts and steel girders. The roadway was also narrowed over a portion of this north direction driveage. This section was the most stable of the I South/2 South driveage and remained so until the steel girders were removed from the test section.

3.3 ROOF STRATA BEHAVIOUR

The behaviour of the roof strata was monitored during the TMP by roof extensometers, convergence rods, load cells and survey levels. The

data indicates that bed separation in the north-south roadways took place above the bolting horizon, 1.8 m above the mining roof, particularly along the eastern part of 1 South and the central and eastern part of 2 South near the 4 West junction.

This bed separation was not observed in 3 West, 4 West and 5 West. As described in Section 3.2, the deduced stress field is thought to initiate the strain (bed separation) in the north-south heading. The very low tensile strength of the roof strata perpendicular to bedding allows bed separation to continue upwards into the immediate roof with time.

Caving characteristics demonstrated in 1 South test section show the very small particle size in the caved rock mass and the relatively high initial (5 m) cave above a narrow development roadway. This unplanned roof fall indicates that caving characteristics of this roof strata will pose no impediment to longwall mining.

3.4 CONCLUSIONS

The results from the TMP in Kipp Mine indicate the following:

a. Roadways driven east to west, west to east, and in a south-westerly direction parallel to the main cleat exhibited better roof conditions than those driven in a north to south direction.

b. Roof conditions in 2 South were slightly better than those found in 1 South. The east rib of 1 South sheared parallel to the rib line and dropped in the order of 30 cm from its mining horizon and shear fractures were observed along the east rib up to 60 cm above the roof line. The deduced stress field is interpreted to enhance this failure mode.

c. Test stations in 3 West and 4 West indicate that for the short term (months), roof bolts (Rebar and Swellex) are capable of supporting the roof.

d. Test Section I South indicates that over a very short term (weeks) roof sag along the east rib was accelerated with the removal of the roof to floor supports, and that roof bolts alone were inadequate. Roof collapse occurred in Test Section I South.

e. Floor heave in the development section was not excessive and was less than 5 cm for roadways 4.5 m and 5 m wide with pillars of coal between 25 m and 35 m between roadways.

f. The mine was very dry in the TMP with the exception of some very limited water seepage in the vicinity of I South/5 West junction and the fault zone in 5 West.

g. Instrumentation installed during the TMP has shown that roof extensometers installed during the initial development will give a good indication of the longer term roof condition after only a short period of time. This indicates that strata conditions could be controlled more easily with available warning. This roof behaviour, in our experience, is quite different from that in Mountain/Foothills mines, where little or no warning is provided to the operator. It would be of great benefit to develop this predictability further.

Associated Mining Consultants Ltd
Golder Associates

74 950 mE

5 514 950 mN

5 514 900 mN

5 514 850 mN

CLEAT

PREVIOUS MINING → TRIAL MINING PROGRAM (HORIZON CHANGE)

1 SOUTH

1 SOUTH

65° to 30°

HORIZON CHANGE

55 m LONG x 3m WIDE
INITIAL CUT LEFT UNSUPPORTED

5 WEST

2 WEST

3 WEST

CLEAT

4 WEST

HORIZON CHANGE

74 900 mE

74 900 mE

PREV. MINING → TRIAL MINING PROGRAM (HORIZON CHANGE)

2 SOUTH

HORIZON CHANGE

2 SOUTH (EXTENSION)

HORIZON CHANGE

LEGEND

Limit of Roof Spalling in excess of 15 cm, prior to installation of set

Overcutting into roof by the Continuous Miner is not shown

Direction of Mining

0 5 10 15 20 25 m

3 - TM GRID COORDINATE SYSTEM
GRID REF DWG No K83-116-A

FIGURE 3 - 1

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

INITIAL ROOF STABILITY PLAN
TRIAL MINING PROGRAM - KIPP

P&T3-0604

PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SECTION 4 - CORRELATION FROM KIPP TO PICTURE BUTTE

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PICTURE BUTTE COAL PROJECT
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

SECTION 4 - CORRELATION FROM
KIPP TO PICTURE BUTTE

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SECTION 4 - CORRELATION FROM KIPP TO PICTURE BUTTE

4.1 PURPOSE OF CORRELATION

The most recent relevant data available concerning mining in the Lethbridge area has come from the TMP at the Kipp Mine. Detailed geologic and geotechnical data must be correlated between Kipp and Picture Butte in order to extrapolate the strata behaviour information from Kipp to Picture Butte.

The initial correlation was made on available geologic data. This data was abundant for the Kipp and the Picture Butte Areas, but very sparse for the intervening Shaughnessy block. General conclusions with regard to predicted roof and floor behaviour during potential mining at Picture Butte were extrapolated from conditions encountered during the TMP at Kipp based on previously accumulated geotechnical data from the Picture Butte area

4.2 GEOLOGIC CORRELATION

A detailed stratigraphic correlation was made between Picture Butte and the Kipp Mine. The lateral continuity of prominent lithological units and their qualitative aspects were assessed to determine the similarities and differences in the depositional sequence between the two areas. This in turn allowed correlation of geologic units exhibiting similar mechanical behaviour and known physical properties.

4.2.1 Scope of Work

Drillhole information acquired during various exploration and geotechnical programs at Kipp, Picture Butte and Shaughnessy between 1975 and 1982 was studied, and preliminary geologic cross-sections constructed. Stratigraphic fence diagrams were also constructed for Kipp (Figure 4-1) and Picture Butte (Figure 4-2), and these subsequently 'linked' by a third fence diagram across the Shaughnessy Area (Figure 4-3). In constructing the fence diagrams only a limited number of representative drillholes were used.

The Kipp and Picture Butte fence diagrams were constructed by plotting an isometric grid on which drillholes were located by means of their co-ordinates. The true location of some Picture Butte boreholes had to be shifted for reasons of clarity. In the Kipp fence diagram overcrowding of borehole profiles was avoided by increasing the north/south scale by 50% with respect to the east/west axis. For the Shaughnessy fence diagram true UTM co-ordinates were used to locate boreholes on a true grid system.

The lithological interpretation was presented for each borehole in each of the fence diagrams together with its density log, with the exception of BH W-10 for which only an electrical resistance log was available. The geophysical logs had varying scales, and were photographically reduced (or enlarged) to a common scale.

4.2.2 Criteria

The most significant geologic parameters were identified, and used for the correlation, using the following criteria:

- continuity of the different coal horizons that comprise the Galt Seam.
- relative position of the Main Rider Seams.
- persistance of and facies changes within the Ostrea Shell Zone.
- location of bentonitic horizons, particularly in the immediate floor horizons.
- persistance of ferruginous siltstone bands.

Lithologs and their variations were established on the basis of their gamma, formation density, and where available, focussed resistivity responses. The thicknesses of the various strata were mainly determined from the focussed beam logs. In many cases, these also enabled a distinction to be made between carbonaceous mudstones and mudstone/coal laminae, as in BH 75-19, BH 75-20, BH W-7 and BH W-12. In some cases study of the gamma ray logs revealed the presence of bentonitic strata (indicated by a shift of some 10 to 20 API in the gamma ray flux).

In general terms, the different lithologies, and their response on the logs, were taken as follows:

	DENSITY	RESISTIVITY (ohm.m)
Coal	Less than 1.7	Greater than 100
Argillaceous Coal	1.70 - 1.85	60 - 100
Carbonaceous Mudstone	1.85 - 2.00	40 - 60
Carbon Bearing Mudstone	2.00 - 2.35	35 - 40
Mudstone	2.35 - 2.50	Less than 35
Siltstone	Greater than 2.5	30 - 35

These criteria were developed with reference to core descriptions when available, or from those of nearby boreholes. In the

interpretation these values were used only as a general guide and were not strictly adhered to.

The correlation was made in the three areas taking the base of the Galt Seam, as recognized in Kipp, as the isochronic reference surface. It should be pointed out that the Galt Seam at Picture Butte includes coal splits below this reference horizon.

4.2.3 The Kipp Area

The fence diagram for the Kipp area is shown on Figure 4-1, and the typical seam section on Figure 4-4.

a. Roof Strata

The roof sequence of the Galt Seam was correlated up to the Oldman-Bearpaw transition zone. The stratigraphic sequence in descending order to the Galt Seam comprises:

- Mudstone (Oldman and Bearpaw Formations) — Main Roof Strata
- Upper Rider Seam
- Mudstone
- Lower Rider Seam — Main Rider Seam
- Mudstone
- Siltstone
- Carbonaceous mudstone to coal
- Silty Mudstone Band (Oztrea Shell Zone equivalent)
- Mudstone
- Galt-Rider (Argillaceous Coal)
- Mudstone
- Galt Seam — Immediate Roof Strata

The total thickness of this sequence to the top of the Main Rider Coal Seam (Upper Rider Seam) remains fairly constant, with a general thinning of the sequence from 7.6 m (BH 78-5) in the southwest, to 6.7 m (BH 75-22) in the northeast. In the immediate Kipp mine area this sequence is approximately 7.0 m thick. The relative position of the different units within the sequence is also fairly constant, the only variables are some facies changes.

The thickness of the mudstone band immediately above the Galt Seam generally decreases in a northerly direction across the area from a maximum of approximately 0.7 m in the southwest to approximately 0.3 m in the northwest.

The Galt-Rider Seam, which occurs immediately above the roof mudstone band, consists of coal to argillaceous coal and is a constant feature across the Kipp area. This thin coal seam usually has a thickness of approximately 0.35 m, although in the mine itself it appears to decrease in thickness to 0.18 m.

The Ostrea Shell Zone does not appear at Kipp, but its horizon is represented on gamma and density logs, and is interpreted as being represented by a compact, cemented, silty mudstone with arenaceous laminae.

A carbonaceous mudstone to argillaceous coal band is located approximately 1.8 m to 2.0 m above the top of the Galt Seam and correlates well across the area. Overlying this unit is a 0.25 m thick, hard, competent, dark grey siltstone band, with occasional fine sandy laminae. The density log of BH 79-8 indicates that two such siltstone bands exist.

At the top of the sequence is the Main Rider Seam which comprises two coal seams, namely the Upper and Lower Rider, separated by a mudstone band. In the mine area the Lower Rider consists of argillaceous coal some 0.3 m thick. The Upper Rider Seam is between 0.4 m and 0.5 m thick. In the southwest and central area the Upper Rider Seam dominates whereas toward the northeast, the Lower Rider Seam tends to dominate. In BH 75-22 the Upper Rider occurs as a carbonaceous mudstone. Similarily, the Lower Rider in BH 75-3 also appears to occur as a carbonaceous mudstone. In BH 78-5 the Lower Rider appears to be absent from the roof sequence.

The Oldman-Bearpaw transition zone throughout the area appears gradational and is generally composed of mudstones. The Bearpaw Shales occur as dark grey, massive mudstones and shales with occasional bentonitic partings. The uppermost Oldman Formation Strata has a more varied lithology.

b. The Galt Seam

The overall thickness of the Galt Seam at Kipp as encountered in boreholes varies from between 1.49 m (BH 75-22) and 1.89 m (BH 75-2) with an average thickness of 1.65 m (the thickness encountered in the Kipp Mine). According to the Norwest 1979 study, the thickness of the Galt Seam varies from 1.28 m to 1.89 m across the Kipp lease area. Over relatively short distances there are small variations in thickness which range between 6 cm and 9 cm, as exemplified by comparison of closely spaced boreholes (BH 79-8, BH 79-9, BH 79-10 and BH 79-11).

The Galt Seam at Kipp consists of three coal bands split by two partings. The basic geophysical signature of the seam is consistant across the area, with the two partings occurring at predict-

able locations within the seam section. The main variation within the seam section occurs as an occasional thinning of the lower parting, from a maximum thickness of approximately 0.3 m to almost zero, as indicated in BH 75-2 and BH 78-7.

c. Floor Strata

Between 0.05 m and 0.20 m below the base of the Galt Seam the lithology grades from carbonaceous mudstone to mudstone. A slightly strong, 0.3 m thick silty mudstone/siltstone band is located approximately 1 m below the seam, except in the southwest part of the property. Coal seams ranging in composition from argillaceous coal to carbonaceous mudstone are located within the immediate 2.0 m of floor strata and bentonitic bands within 2.0 m of the base of the Galt Seam. Lower down in the floor sequence the lithology grades into wavy bedded arenaceous siltstones and sandstones.

BH 79-8, BH 82-1 and BH 82-3 intersected the coarser sequence approximately 2.4 m higher than anticipated which may be due to pre-Galt faulting and erosion (Norwest 1981).

4.2.4 The Shaughnessy Area

Only three small scale geophysical logs (BH W-10, BH W-7 and BH W-12) were available along a northeast/southwest line connecting the Picture Butte and Kipp areas. However, a number of general trends were identified (Figures 4-3 and 4-4).

a. Roof Strata

The dominance of the Lower Rider Seam in a northeasterly direction across Kipp continues in the Shaughnessy area. The Upper Rider Seam 'pinches' out and is absent north of BH W-12.

The Ostrea Shell Zone is absent in BH W-7, and occurs in BH W-12. The appearance of the shell bed between these boreholes coincides with the absence of the siltstone band which exists across the Kipp Area.

b. Galt Seam

The Galt-Rider Seam which occurs in the Kipp Area merges with the Galt Seam between BH W-10 and BH W-7. The carbonaceous mudstone that occurs above the Galt-Rider in the Kipp Area undergoes a facies change to coal, and also appears to merge with the Galt Seam.

The Galt Seam section here has a total thickness of between 1.8 m and 2.7 m and appears to contain only one major mudstone parting.

c. Floor Strata

At Shaughnessy the floor strata consist of mudstones and thin coal seams which are separated and underlain by a similar mudstone to that at Kipp and at Picture Butte (Figure 4-4). In BH W-7 and BH W-12 bentonitic horizons were identified within these mudstone units, as was occasionally the case in the Kipp Area.

4.2.5 The Picture Butte Area

The fence diagram constructed for the central region of the Picture Butte lease area is shown in Figure 4-2, and a typical seam section in Figure 4-4.

a. Roof Strata

Consistent with the interpretation in the Kipp Area, the roof of the Galt Seam was correlated to just above the Main Rider Seam. At Picture Butte the total thickness of this sequence to the bottom of the Main Rider varies from 3.0 m to 4.0 m. The stratigraphic sequence in descending order comprises:

- Mudstone (Oldman and Bearpaw Formations)
- Main Rider Seam
- Mudstone
- Ferruginous Siltstone/ Concretionary Ironstone
- Mudstone
- Ostrea Shell Zone
- Mudstone
- Galt Seam

} Immediate Roof Strata

In contrast to the sequence encountered at Kipp, no carbonaceous horizons occur between the Galt Seam and its uppermost, and only, Rider Seam, the sedimentary sequence here appearing to be slightly more uniform.

In BH PB 75-23 and BH PB 80-10 the Ostrea Shell Zone was encountered. This unit appears to be absent in the eastern half of the study area.

In most available core descriptions the 0.4 m thick siltstone band, which generally occurs between the Galt Seam and the Main Rider

Seam, was described as ferruginous. The density logs infer that this unit probably grades locally into a concretionary ironstone band (BH PB 81-13, BH PB 81-31, BH PB 81-25). The siltstone/-ironstone band appears to have pinched out completely west of BH PB 75-23 and BH PB 75-19. As shown on the fence diagram, its presence appears to alternate with the presence of the Ostrea Shell Zone.

b. Galt Seam

The Galt Seam at Picture Butte comprises four coal bands, the uppermost band referred to as Split #1 in Norwest's 1980 Report, being equivalent to the Galt Seam as recognized at Kipp. This main seam is underlain by a series of coal seams referred to by Norwest as Split #2, #3 and #4 which are separated by mudstone bands (Figure 4-4).

The maximum proven thickness of the total Galt Seam section at Picture Butte is 3.8 m. However, the Galt Seam as it appears at Kipp correllates with the Split #1 (or Main Seam) as recognized at Picture Butte. The thickness of the Split #1 Seam across the Picture Butte area varies from approximately 1.5 m to 1.8 m with a gradual increase in a north-westerly direction. In the north of the study area a prominent mudstone parting is located near the top of the seam and varies in thickness from 0.2 m to 0.3 m (BH PB 81-14, BH PB 81-31, BH PB 81-13). The thickness of this parting diminishes toward the south (BH PB 81-14, BH PB 75-23, BH PB 80-10 and BH PB 75-19) where it is replaced by argillaceous coal. A thinner parting near the base of the seam is more erratic in its presence and composition.

c. Floor Strata

The immediate Split #1 Seam floor strata consists of a mudstone band ranging from a few centimetres up up to 2 m thick with 1 to 2 m in the limited area under consideration, (referred to by Norwest 1980 as Band #1), which is underlain by a thin coal seam (Norwest's Split #2) which is in turn underlain by an interlayered sequence of mudstone and thin coal seams (Norwests' S3 and S4). The floor strata encountered at Picture Butte will depend on the economically mineable seam section. However, whichever splits are mined, the floor will be established on mudstones.

In BH PB 81-28 only the two lower coal seams were identified, being separated by a carbon bearing mudstone. Locally, all the thin coal seams in the floor strata may be absent, indicating post depositional erosion (BH PB 81-25).

Information available does not indicate the presence of bentonitic horizons in the Galt Seam floor strata at Picture Butte.

4.2.6 Conclusions

The lithologic correllation across the Kipp/Picture Butte area enables several conclusions to be made. The main points are as follows:

a. At Picture Butte the Galt Seam comprises four coal bands, the uppermost band being equivalent to the Galt Seam as recognized at Kipp.

b. The mudstone parting between the Galt Seam and the Galt-Rider at Kipp is significantly thinner at Picture Butte, and the Galt Seam as defined at Picture Butte includes the

Galt-Rider Seam. It follows that there is no coal in the immediate roof at Picture Butte.

c. It is not possible to confidently correlate the interseam partings at Kipp and Picture Butte either on a location or lithologic basis. The parting near the top of the Galt Seam at Kipp (as used to define the roof in parts of the Kipp TMP) comprises different material than the partings at Picture Butte. Also, the parting near the top of the Galt Seam at Picture Butte is considerably thicker than the parting near the top of the Galt Seam at Kipp.

d. At Picture Butte the immediate roof sequence between the Galt Seam and the Main Rider is considerably thinner than at Kipp.

e. At Kipp the Main Rider consists of two seams, approximately 6 m to 8 m above the Galt Seam. At Picture Butte only one rider seam exists and is approximately 3 m above the Galt Seam.

f. The Ostrea Shell Zone that is occasionally present immediately above the Galt Seam at Picture Butte does not exist at Kipp, although it does appear to be represented by a silty mudstone band.

g. From geologic descriptions, geophysical responses, and from visual observations, the mudstone immediately below the Galt Seam at Picture Butte appears to be similar to that encountered at Kipp.

The above geologic parameters have been used in the evaluation of geotechnical data accumulated at Kipp with respect to the anticipated mining conditions at Picture Butte.

4.3 GEOTECHNICAL CORRELATION

Observations made during the TMP at Kipp, together with core logging, detailed laboratory testing and the results of previous geotechnical studies, have all been used to correlate geotechnical parameters between Kipp and Picture Butte. The similarities and/or differences have been identified and are discussed so that data accumulated from future geotechnical studies at Picture Butte can be objectively considered in the light of the mining conditions experienced at the Kipp Mine.

4.3.1 Roof Strata

Bedding break intervals recorded for the immediate roof in the Picture Butte area generally range from 0.05 m to 0.3 m (BH PB 81-24 to BH PB 81-29) and can be classified as being closely spaced. The generally wider bedding break spacing recorded in core from the Kipp Mine is probably attributable to the superior core recovery obtained from the underground diamond drilling.

RQD values recorded for the immediate roof strata at Picture Butte are generally 90%, except where in close proximity to the Galt Seam. This is in agreement with observations made at Kipp, and indicates that the absence of pronounced jointing in the exposed roof at Kipp will also likely be encountered at Picture Butte.

Both core and bulk samples from the roof strata at Kipp had compressive strength values of approximately 50 MPa. Previously recorded values by Dames and Moore (1979) were approximately 35 MPa. Results of uniaxial compressive strength tests performed on roof mudstone samples from Picture Butte (Table 4-1) indicate compressive strengths varying from approximately 9 MPa to 14 MPa. Since both previous surface drilling programs were performed with the same drilling equipment (Ingersol Rand TH60 with air flush), this would suggest that the roof strata at Picture Butte could be weaker than at Kipp. Further strength testing over a wider area would be required to test this observation.

Atterberg limits determined on a typical Bearpaw mudstone sample from Picture Butte indicate a 'clay-rock' of medium plasticity. Similar results were obtained on immediate roof samples (Oldman Formation) from the Kipp Mine. The calcite content in roof strata at Kipp is low (less than 1.5%). Similarly, calcite contents in the mudstone roof strata in the central Picture Butte area are also low (generally less than 0.6%), but slightly higher for Bearpaw mudstones (less than 1.2%). These test results indicate a minimal contrast between the mudstones of the Oldman and Lower Bearpaw Formations, and suggest that the mudstones of both formations should react similarly to mining.

4.3.2 Galt Seam

Within the central Picture Butte area, bedding break intervals in the Galt Seam recorded in BH PB 81-24, BH PB 81-25, BH PB 81-26, BH PB 81-27, BH PB 81-28 and BH PB 81-29 generally range from 0.05 m to 0.6 m (close to moderately close). Values obtained for bedding break intervals by Dames and Moore (1979) in the Kipp area generally varied between 0.04 m to 0.1 m. RQD values documented from core from the above boreholes

range from 25% to 95%, compared with a range of 50% to 75% recorded by Dames and Moore (1979) at Kipp. This indicates that the structural discontinuities (bedding, cleat, etc.) within the Galt Seam at Picture Butte should be the same as those encountered at Kipp.

The unconfined compressive strength recorded for the Galt Seam coal at Picture Butte (Table 4-1) is approximately 17 MPa, which is 70% higher than the equivalent values at Kipp.

4.3.3 Floor Strata

Bedding break intervals recorded for the central Picture Butte area show a great variability (0.01 m to 0.35 m). The values measured in BH 82-1 and BH 82-3 at Kipp are generally higher (0.1 m to 0.5 m). RQD values at Picture Butte vary generally between 50% and 75% and are in general agreement with observations made at Kipp by both Golder and Dames and Moore.

The uniaxial compressive strength values for the various floor strata are shown in Table 4-1. Strength of the siltstone of approximately 40 MPa is in close agreement with results from the underground drilling program at Kipp. The strength of the sandstone is approximately 35 MPa, which is close to the range of 21 MPa to 33 MPa previously recorded by Dames and Moore. Sandstone samples recovered from diamond drilling at Kipp exhibited strengths of almost 55 MPa.

The general observation made for Kipp is that no significant difference in strength exists between mudstones and carbonaceous mudstones in the seam floor. However this is not the case at Picture Butte, where a wider range of strength values was found for carbonaceous mudstones (from 1.5 MPa to 22 MPa) compared

to the strength of the mudstones (7.5 MPa to 22.8 MPa). The highest value is substantially lower than the highest value recorded by Dames and Moore at Kipp (37 MPa). The same drilling equipment was used during both geotechnical programs (Dames and Moore 1979, Golder 1981) and it is therefore reasonable to assume that the strength of the mudstone in the floor at Picture Butte is less than at Kipp, although it should be noted that the data from Picture Butte was obtained from a very limited area.

4.3.4 Slake Durability of Mudstone

Comparison of slake durability test results (second cycle) from Kipp (38% in 9 tests, Golder 1982) with results from Picture Butte (0%, 0%, 58%, Golder 1981, see Table 4-2) indicates that the resistance to slaking of the mudstones at Picture Butte will probably be less than at Kipp. However, it is possible that the test samples from Picture Butte were adversely affected by the surface drilling, whereas samples from Kipp were relatively undisturbed.

4.3.5 Clay Mineralogy

From the surface drilling program to be carried out in 1983 it may not be possible to carry out spectrographic analyis of samples from Picture Butte because of sample pollution by bentonite-based drilling mud. It is therefore assumed at the present time that analyses of samples from Kipp will equally apply to Picture Butte.

The results of spectrographic analyses on samples from the Kipp Mine indicate stability of mine openings will be dictated by both minerological and physical controls. The high montmorillonite

content is significant, and the type of montmorillonite (Sodium or Calcium) may have important geotechnical ramifications. The presence of sodium montmorillonite has a great influence on the strength and behaviour characteristics of clay shales, particularly if a large degree of fissility exists.

Montmorillonites generally have very high liquid limit values. It will therefore be essential that moisture levels be maintained at a minimum level to prevent ground control problems caused by swelling and desiccation of floor and roof strata respectively.

4.3.6 Conclusions

The conclusions derived from the geotechnical correlation can be summarized as follows:

a. The frequency of bedding break discontinuities appears higher at Picture Butte than at Kipp.

b. Although the mudstones of the Oldman and Lower Bearpaw Shale Formations have similar material properties at both Kipp and Picture Butte, the roof strata at Picture Butte appears to be weaker than at Kipp.

c. Laboratory test data indicated that the unconfined compressive strength of the Galt Seam at Picture Butte is higher than at Kipp.

d. Based on data at Picture Butte confined to a limited area the strength of the mudstones in the floor strata at Picture Butte appears to be less than at Kipp.

e. Laboratory test results indicate that the mudstones at Picture Butte will probably have less resistance to slaking (mechanical breakdown) than the mudstones exposed at Kipp. Further testing is required to confirm this.

4.4 EXTRAPOLATION OF MINING EXPERIENCE TO PICTURE BUTTE

4.4.1 Roadway Conditions

In general it was found at Kipp that a coal roof resulted in better roof conditions, with less roof spalling and bed separation than when a shale roof was attempted. Even though roadway orientation and direction of drivage had an effect on the roof conditions it is recommended that all rectangular roadway drivage at Picture Butte be done under a coal roof. This is particularly important as the limited available geotechnical data indicates weaker mudstones in the Picture Butte area than encountered at Kipp.

The experience of extensive roof support changes and repairs at mining horizon changes indicates that similar horizon changes at Picture Butte should be given more support with stronger beams or closer support spacing.

The coal at Kipp was found to be strong and did not give sloughing problems. Rib pressure on set posts was noticed on 2 to 3 sets only at the I South/5 West junction and this condition was taken to be localized in nature. It is assumed from these observations that there will be no rib pressure problems at Picture Butte. The small amount of rib coal sloughing that did occur was in 3 West which was oriented along the cleat direction. Similar sloughing may occur in roadways oriented along the cleat in Picture Butte.

Geotechnical data comparisons of mudstone strength parameters between Kipp and Picture Butte indicate lower slake durability and compressive strength in the Picture Butte area. This may present a problem with trackless equipment, especially if water is present. Since floor heave was non-existent in all headings except I South, which was measured at less than 5 cm, rail transport for men and materials and belt transport of development coal via bridge conveyors could be considered for any future mine planning in the Picture Butte area.

4.4.2 Mining Conditions

The only appreciable quantities of water encountered at Kipp came from water line breaks in headings. This water on the floor resulted in posts sinking into the floor, requiring extensive repairs to the sets. If the geotechnical studies to be carried out at Picture Butte indicate the presence of water near the Galt Seam it will have to be dealt with in the mine design.

In general it was found that secondary services of water from dust suppression on the continuous miner and from stoper bolting operations, did not contribute to floor degradation at Kipp. Weaker mudstone indicated by lower compressive strengths and slake durabilities of Picture Butte mudstones may be more sensitive to water.

Methane was detected only in small quantities at Kipp and therefore the same condition is expected at Picture Butte.

TABLE 4-1

PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

TEST RESULTS - UNIAXIAL COMPRESSIVE STRENGTH

SAMPLE #	DRILLHOLE #	DEPTH (m)	DISTANCE ABOVE (+) OR BELOW (-) GALT (m)	UNIAXIAL COMPRESSIVE STRENGTH (MPa)
Galt Coal				
4	PB81-25	85.6	-	17.6
10	PB81-26	83.9	-	17.9
23	PB81-29	81.7	-	14.7
Mudstone (Roof - Oldman)				
11	PB81-27	81.9	+3.9	9.2
21	PB81-29	80.8	+0.3	10.8
9	PB81-26	82.2	+1.1	9.5
28	PB81-29	79.9	+1.2	10.2
12	PB81-27	84.1	+1.7	9.0
20	PB81-29	78.8	+2.3	6.6
1	PB81-24	75.8	+3.2	13.5
19	PB81-29	77.7	+3.4	8.9
Mudstone (Roof - Bearpaw)				
3	PB81-24	79.9	+4.5	6.1
2	PB81-24	77.6	+6.8	5.8
11	PB81-26	76.3	+7.0	9.2
18	PB81-29	76.$	+4.7	8.6
Ferrugenous Siltstone (Roof)				
26	PB81-29	79.7	+1.4	20.8
26	PB81-24	77.2	+1.8	20.8
Carbonaceous Mudstone (Floor)				
13	PB81-27	89.7	-0.1	1.5
14	PB81-27	90.0	-0.4	4.6
24	PB81-29	82.8	-0.2	22.0
Mudstone (Floor)				
25	PB81-29	83.4	-0.3	7.5
22	PB81-29	83.8	-0.8	13.1
Siltstone (Floor)				
6	PB81-25	87.1	-1.2	40.3
10A	PB81-26	86.3	-1.5	22.8
Sandstone (Floor)				
15	PB81-	90.9	-1.2	34.5

TABLE 4-2

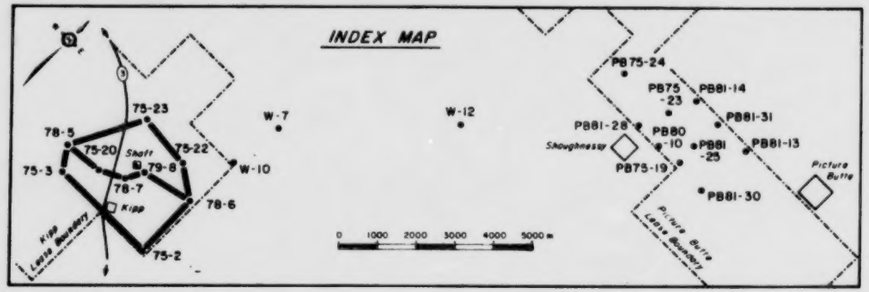
PICTURE BUTTE COAL PROJECT

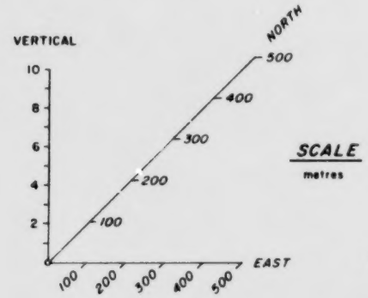
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

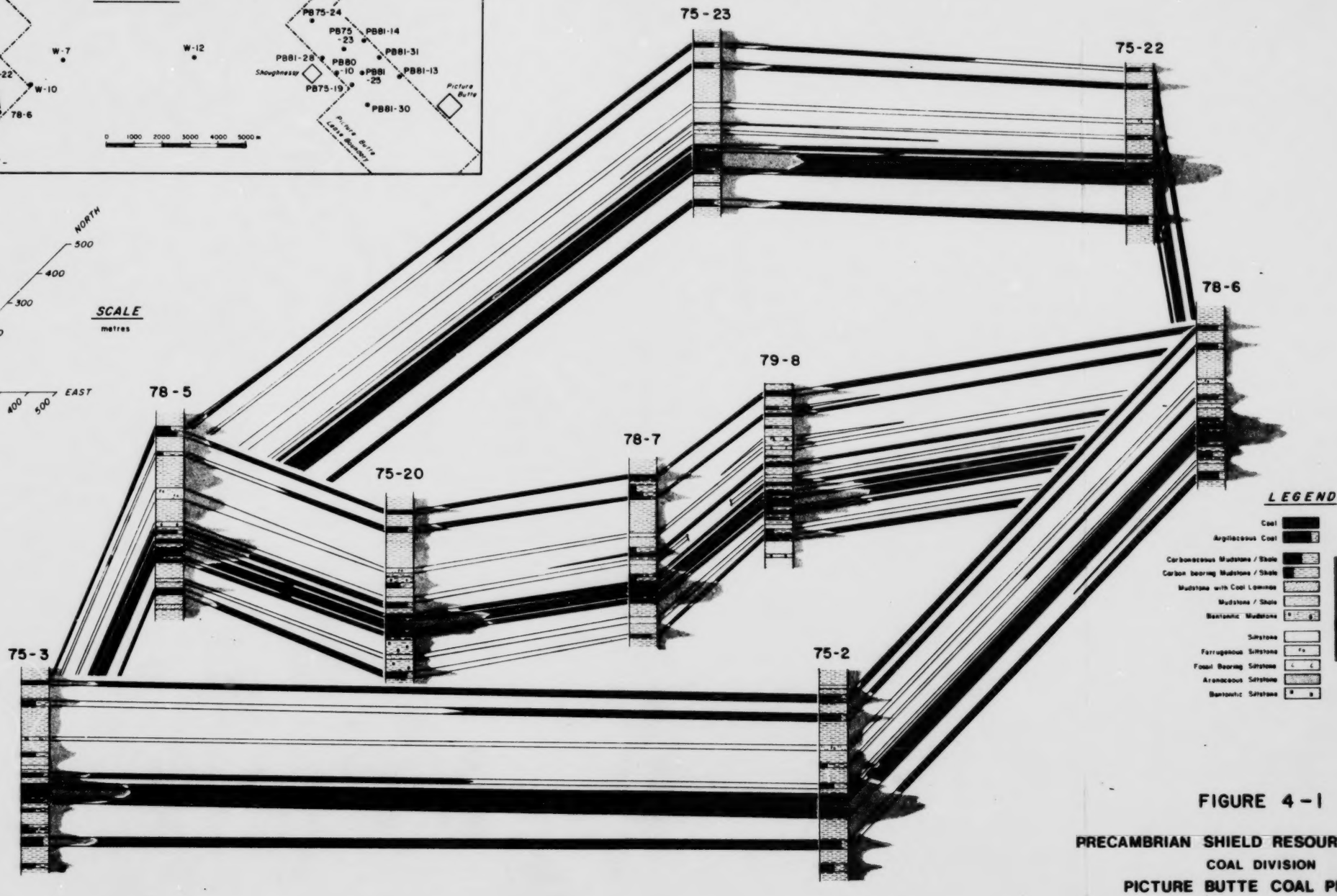
TEST RESULTS - SLAKE DURABILITY, CALCITE CONTENT, MOISTURE CONTENT, AND ATTERBERG LIMITS

SAMPLE #	LOCATION	DRILL HOLE #	DISTANCE ABOVE (+) OR BELOW (-) GALT (m) From	To	SLAKE DURABILITY (% RETAINED) 1st CYCLE	2nd CYCLE	CALCITE CONT. (%)	ATTERBERG LIMITS LIQUID (%)	PLASTIC (%)	PLAST. INDEX (%)
Mudstone (Roof - Bearpaw)										
2	PB 81-25	78.1	+6.3					41	24	17
17	PB 81.29	50.3	+30.8				1.0			
	ST 3	55.7					1.2			
	ST 3	54.6					1.0			
	ST 3	53.1					1.1			
Mudstone (Roof - Oldman)										
21	PB 81-29	80.8	+3.0				2.4			
9	PB 81-26	81.9	+1.4				0.6			
26	PB 81-24	77.2	+1.8				0.3			
11	PB 81-27	81.9	+3.9				0.3			
Gouge (Roof)	PB 81-25	81.8	+2.6					67	30	37
Carbonaceous Mudstone (Floor)										
5	PB 81-25	86.0	-0.1		68	32				
24	PB 81-29	82.9	-0.2		83	58				
13	PB 81-27	89.7	-1.5		70	27				
Mudstone (Floor)										
25	PB 81-29	83.4	-0.8		28	0	0.1	73	39	34
22	PB 81-29	83.8	-1.2		82	58	0.2	52	21	31
14	PB 81-27	89.9	-2.9		0	0		76	22	54
Siltstone (Floor)										
10A	PB 81-26	86.2	-1.5		96	93	2.1			

Associated Mining Consultants Ltd.
Golder Associates







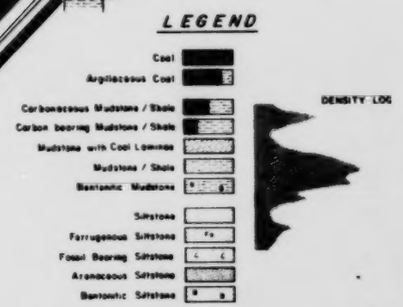


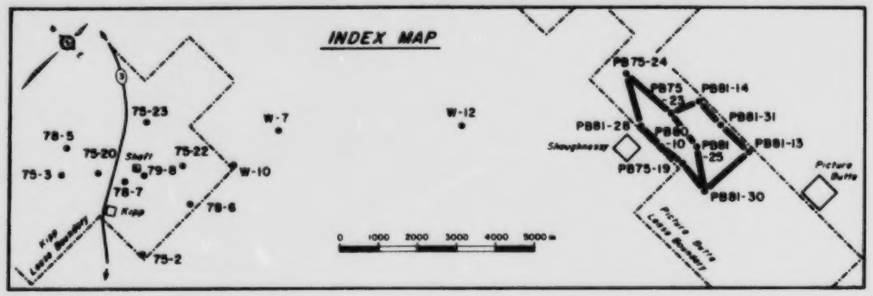
FIGURE 4-1

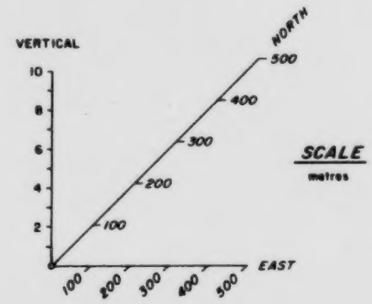
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

FENCE DIAGRAM
KIPP

PA73-003





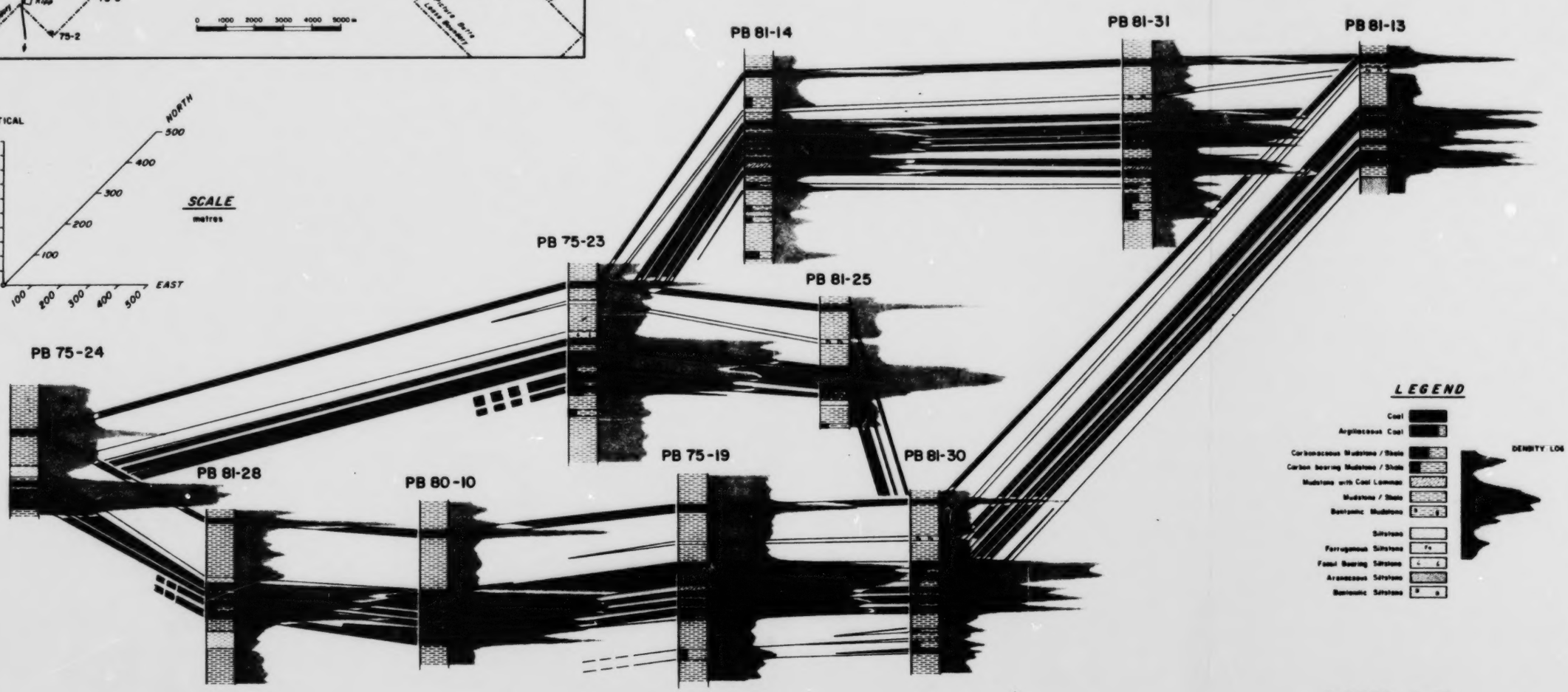


FIGURE 4-2

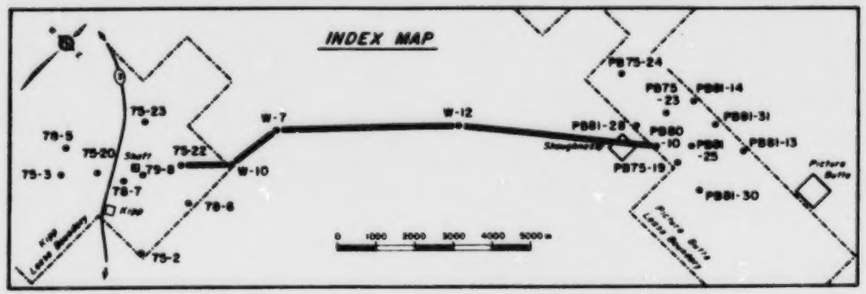
PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

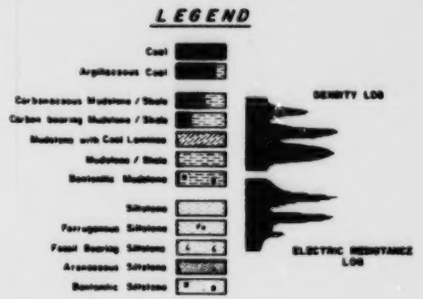
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

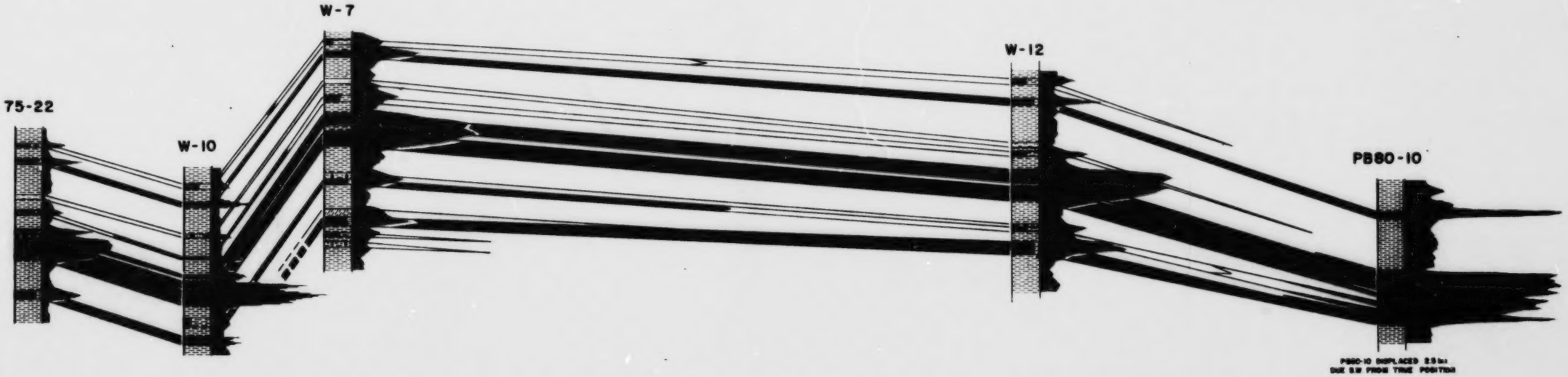
FENCE DIAGRAM
PICTURE BUTTE

PA73-002

Associated Mining Consultants Ltd.
Golder Associates







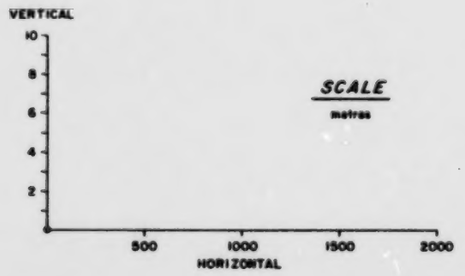


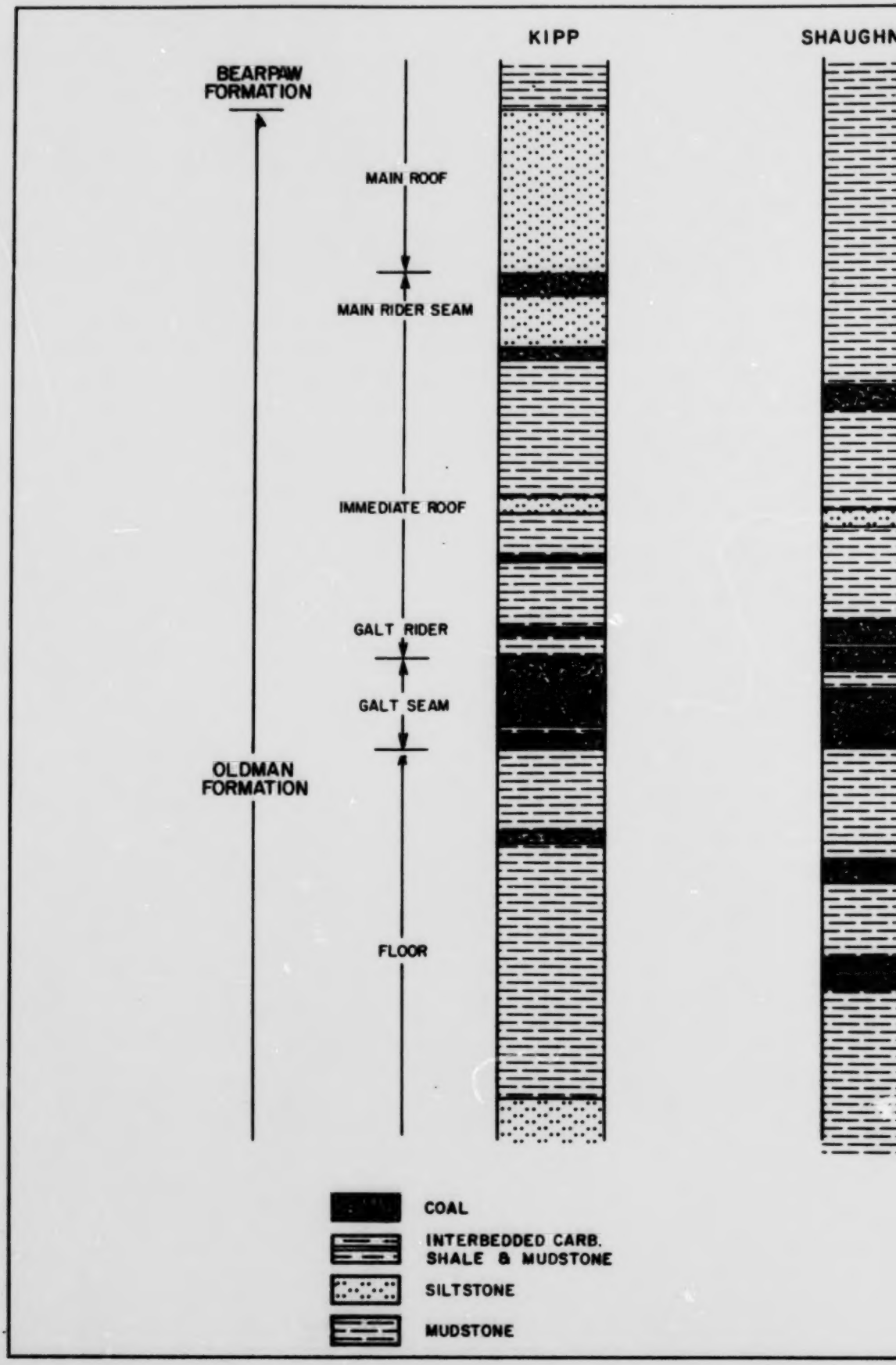
FIGURE 4-3

PRECAMBRIAN SHIELD RESOURCES LIMITED
COAL DIVISION
PICTURE BUTTE COAL PROJECT

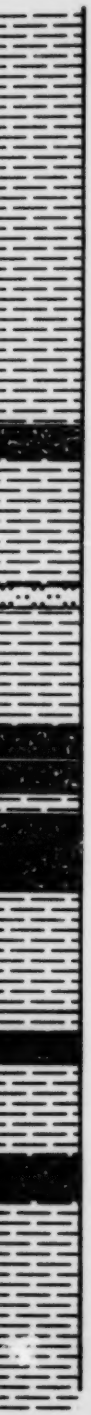
A TECHNICAL ASSESSMENT OF PICTURE BUTTE

FENCE DIAGRAM
KIPP TO PICTURE BUTTE

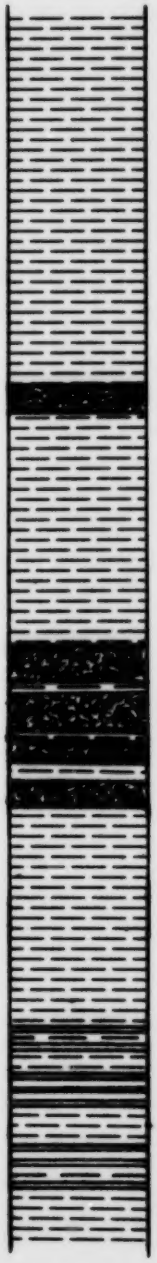
PA75-004

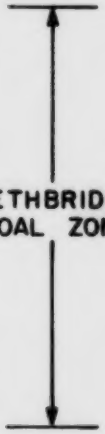

KIPP
SHAUGHN
BEARPAW FORMATION
MAIN ROOF
MAIN RIDER SEAM
IMMEDIATE ROOF
GALT RIDER
GALT SEAM
OLDMAN FORMATION
FLOOR
COAL
INTERBEDDED CARB. SHALE & MUDSTONE
SILTSTONE
MUDSTONE

GHNESSY



PICTURE BUTTE





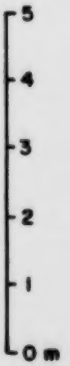


FIGURE 4 - 4

PRECAMBRIAN SHIELD RESOURCES LIMITED

COAL DIVISION

PICTURE BUTTE COAL PROJECT

A TECHNICAL ASSESSMENT OF PICTURE BUTTE

GEOLOGIC SECTIONS

KIPP - SHAUGHNESSY - PICTURE BUTTE

SECTION 5 - UNDERGROUND MINING AT PICTURE BUTTE

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SECTION 5 - UNDERGROUND MINING AT PICTURE BUTTE

5.1 GENERAL

Underground mine design is site specific - there is no simple, standard design. Assumptions or deductions made about relevant design criteria must be carefully weighed - if too optimistic then planned production goals are not achieved; if too pessimistic the mine is never developed.

The constraining factors considered for mine design are of two types: fixed parameters (eg: geology, water, gas, old workings, etc.) and variable parameters (eg: design of roadways, mine plan, support systems, services, etc.).

This section deals with the impact of the 'fixed' parameters as determined in this study on the variable design parameters at Picture Butte.

5.2 GROUNDWATER

5.2.1 General Hydrogeologic Parameters at Picture Butte

Large quantities of ground water flow into the mine would be detrimental to the mining operation. As a result, the potential for mine inflows will influence face design and orientation. Water released into the mine from the coal seam or overburden could have a detrimental effect on the behaviour of the exposed floor rocks at Picture Butte, and could restrict the rate of longwall advance.

Available ground water data in the Kipp-Picture Butte area is sparse and very conjectural. Consequently, until more information is obtained with regard to hydraulic conductivities of the various strata, only general comments can be made, and have been based upon data from existing piezometer installations, and hydrological reports of the Lethbridge area.

A pre-glacial buried channel with associated highly permeable Saskatchewan gravel and sand deposits is known to exist to the north and east of the Picture Butte lease area, but does not traverse the PSRL property. The existence of any buried channels or aquifers lying above the Galt Seam across the Picture Butte property is considered extremely unlikely. At Picture Butte, the permeability of the glacial overburden will be very low, in the order of 1×10^{-10} m/s. Isolated gravel bands may have higher permeability, but the water in them will be confined by almost impermeable materials. The anticipated low permeability of the overlying bedrock should minimize any ground water flow from the bedrock/glacial overburden contact into development roadways. Above the extracted longwall panels, mining will result in caving and fracturing of the roof strata. However, even if the fractures induced by mining penetrate the bedrock/glacial overburden contact, it is not anticipated that inflows will pose serious problems because of the clayey/silty nature of the till, and its expected very low permeability.

It is probable that the Galt Seam, with its well-defined cleat system, has a significantly higher permeability than the enclosing strata. Previously collected hydrogeologic data suggested that the Galt seam at Kipp was water bearing, and that is was under considerable pressure, indicating a 'confined aquifer' condition. However, water was not encountered when the Kipp shaft penetrated the seam, and it is significant that water seepage problems

were not encountered during recent trial mining. Also, there are no records of excessive inflows in the Galt No. 8 Mine, and no water wells are known to exist at Picture Butte. It is reasonable to assume that the overall ground water regime at Picture Butte will be similar to Kipp, and that the hydrologic characteristics of the Galt Seam will be similiar. The lack of ground water flow through the coal seam, or from the overlying strata in the test mine, is therefore significant.

The significance of these anticipated hydrogeologic parameters to mining at Picture Butte can only be fully evaluated when proper piezometer installations are made at Picture Butte, and subsequent monitoring and permeability testing has been carried out. As part of the overall feasibility study, pumping tests may be required over the full section of the overburden, and would enable a more detailed assessment of potential inflows, and to confirm whether or not the overlying glacial material should be considered as a constraint to mining and/or shaft sinking at Picture Butte.

5.3 FLOOR CONDITIONS

The physical properties of the floor strata play a very important part in the choice of longwall face equipment and method of roof support in roadways for a successful mining operation.

The results of slake durability and free swelling tests indicate that the swelling and deterioration of immediate floor strata if exposed to wet conditions over a period of time could cause operating problems, both at the working face and in roadways.

The geologic correlation indicates that whichever seam splits are mined at Picture Butte, the floor will be established on mudstone similar to that encountered at Kipp. Laboratory test results indicate that the

plastic limit of these mudstones is much higher than the in situ material moisture contents. We anticipate that the hydrogeologic conditions at Picture Butte will be the same as at Kipp and the floor strata at Picture Butte should, therefore, behave in the same manner as during the TMP.

5.4 ROOF STABILITY

Roof stability in general refers to the stability of the immediate roof and the time duration that it may be left unsupported, either in roadways or at the operating face. There are many factors that contribute to roof stability (mining height/strata properties/structural discontinuities, etc.)

To determine the required face support requirements, it is necessary to estimate the thickness of the immediate roof being that portion of the roof that detaches from the roof strata, deflects downward and requires support as soon as the underlying coal is extracted. As longwall face supports only take stresses imposed by the immediate roof (and not the entire overburden sequence) it is only this zone which requires support. In Britain, the thickness of the immediate roof is usually estimated to be twice the seam height, based on the assumption that when 'caved', the roof rock expands to 50% more than its original volume. However, any deviations from this assumed 'bulking factor' will affect the predicted height of the immediate roof. The caving characteristics of the roof are influenced by the local geologic sequence, and caving above the Galt Seam will probably not stop at twice the extracted seam height. The mudstone/shale roof will shear behind the supports, above the Galt Seam at Picture Butte and together with inherent planes of weakness, will control the size and shape of the caved fragments.

Whilst the height of the roof cave in a roadway does not necessarily reflect the height of a caving zone behind a longwall panel, the nature

of the caved material does indicate how the roof material will behave when caved in gob areas. From the inspection of the caved roof material in the I South roadway at Kipp, a 'bulking factor' of 30% to 35% is more realistic for this type of strata.

The magnitude of abutment pressures caused by cantilevered roof beams depends on the length and thickness of the immediate roof strata that overhangs the gob area behind the face supports. The shale and mudstone that comprise the immediate roof at the Picture Butte property should cave readily into extracted areas a short time after advance of longwall face supports, and the development of long unsupported roof spans in the gob areas that could impose forward abutment pressures ahead of the longwall face are not anticipated.

The absence of joints in the immediate roof strata at Kipp and in drill core from Picture Butte, indicates that periodic roof weighting problems at the face caused by large structurally controlled 'slabs' or 'blocks' in this roof strata will not be experienced.

Caving characteristics can, therefore, be considered as favourable for longwall mining at Picture Butte.

The thickness of bedrock overlying the coal seams is important for both roof stability and face support, and the potential for groundwater inflows into the mine through fractured ground above mined out areas. There is, therefore, a minimum thickness of bedrock overlying the seam that must be considered. The height of the induced 'fracture zone' above mined out panels may vary between 25 and 40 times the extracted seam thickness. If the conservative estimation is assumed, the height of the fracture zone above the thickest anticipated seam extraction (3.8 m) would be approximately 150 m. If the mining section were restricted to 1.5 m, the height of the fracture zone would be approximately 60 m. Similarly, using the lower factor, the height of

the fracture zone would vary between 95 m and 40 m for the respective extracted seam sections. Although there is a large degree of ambiguity in this calculation, it is reasonable to assume that the induced 'fracture zone' will penetrate the bedrock/overburden contact across part of the Picture Butte property.

Records indicate that the glacial material in the area is generally silty, and will have a very low permeability, and it is reasonable to assume that serious water inflow from the glacial overburden will not occur through the fractured roof strata. If the results of pump tests and/or early mining operations indicate that groundwater inflows into gob areas from the glacial overburden could be significant, it may be necessary to limit longwall mining to areas with a thicker bedrock roof section.

5.5 CONSTRAINTS TO EQUIPMENT SELECTION

The expansion and/or physical breakdown of the floor strata will decrease its supporting capacity. The major factors affecting penetration of the supports and conveyor at Picture Butte are the characteristics of the floor strata, and moisture conditions at the face.

Because of potentially poor floor conditions, supports should be selected to provide adequate stability. Floor stress profiles applied to the floor beneath the support should also be considered; if there is extremely high pressure around the tip of the base, the base tends to 'dig' into the floor resulting in decreased roof support capability and also hinders the advance of the support. Pressure distributions of the canopy and base for different supports are available from equipment manufacturers.

A shield-type support is generally recommended for a potentially weak roof, because the horizontal component exerted by the shield canopy tends to push the immediate roof strata toward the face resulting in

more stable conditions. Shield supports are also suitable for seams thicker than 2 m because of their high resistance to tangential forces. With the potential variation in extraction thickness it may be necessary to use more than one size range of face support system throughout the mine.

The selection of face support and handling systems is dependent upon a series of design parameters (support setting load, support yield load, bearing capacity of roof and floor, etc.) and is beyond the scope of this report. Methods of calculating support requirements from rock properties and geologic data are not exact, however several factors affecting face support equipment have been identified during this study, and should be referred to equipment manufacturers when equipment selection studies are undertaken.

5.6 IMPLICATIONS TO MINE LAYOUT

From the studies made thus far, it is not possible to determine if high horizontal stresses exist, and whether or not conditions at Picture Butte will be characterized by a 'good roof' and a 'bad roof' direction, which may be important in obtaining optimum panel and main roadway orientation, and development direction.

Visual observations during the TMP has indicated higher stresses parallel to the cleat, and it is reasonable to assume that this is also the case for Picture Butte. The pronounced cleat will affect the stability of the coal face; coal worked parallel to the cleat will fall off in large slabs, and working a butt cleat will require more energy to cut the coal. By maintaining a north-south, east-west mine layout, a compromise will be attained for both actual mining of the coal and roadway stability. This will result in longwall faces and all roadways being developed at approximately 30° to 60° to the main cleat. At the present time it is not considered that the location and orientation of

gate entries (and therefore panel widths) will be dictated by zones of weakness, joint patterns, or faults, etc. However, a geologic risk analysis should be included with any potential mine plan when an economic assessment is made.

If further studies at Picture Butte indicate that groundwater inflows may be a constraint to mining, the most favourable longwall extraction would be to the east (up-dip), allowing water to drain off into the gob area. In the longwall retreat method of mining, this would involve the development of gate roads in a down-dip direction. Bed separation and induced ground fracturing above roadways will not be so severe as that above the working longwall face, and it is not anticipated that any water problems will be encountered at the face during development of roadways.

5.7 SUBSIDENCE

The land surface is generally used for the production of grain with some cattle grazing, and is traversed by an irrigation system consisting of ditches and canals, as well as by gas pipelines. No major structures exist across the property.

It is generally accepted that variations in the type of stratum between the coal seam and the ground surface do not have any effect on the maximum amount of subsidence. However, the nature of the surficial deposits will affect the shape and surface effects of subsidence caused by mine workings. With the longwall system of mining, the subsidence trough could be expected to have a flat bottomed profile. At Picture Butte the Bearpaw Formation is overlain by up to 100 m of soft overburden deposits. It is anticipated that these deposits will behave in a plastic manner. Consequently, the subsidence profile will be of greater lateral extent than would be expected if bedrock extended to

surface, with gentle boundary slopes without rupture. It follows that the effects of differential settlement will not be so severe.

Number/
Numéro

3

A GEOLOGICAL EVALUATION OF THE TECHNICAL FEASIBILITY OF LONGWALL MINING IN THE GALT SEAM AT KIPP TRIAL MINE NUMBER 1787/E/01 (UP-P-137)

INTERIM REPORT

BY

PRECAMBRIAN SHIELD RESOURCES LIMITED

Work on this project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada

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Volume 82/58-C

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COAL MINING RESEARCH CENTRE
PLATE LOAD TESTS AT KIPP MINE
FOR PRECAMBRIAN SHIELD RESOURCES LTD.
BY
D.H. GREEN AND P.L. WRIGHT
CMRC REPORT 82/58-C OCTOBER 1982

RECEIVED
NOV 25 1982
PRECAMBRIAN SHIELD
COAL DIVISION

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B. CHARTS

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E. PICTURE OF PLATE LOAD TEST EQUIPMENT

F. REFERENCE PAPERS

INTRODUCTION

CMRC was requested to perform plate load bearing tests on floor strata at the Kipp Mine near Lethbridge, Alberta by Precambrian Shield Resources Ltd.

Seven plate load tests were completed, 2 on dry floor coal, 1 on floor coal on which water had been poured, 2 on dry floor shale, 1 on floor shale on which water had been poured, and one overnight test on the floor shale. Precambrian prepared the test site which consisted of a heading 5.8 m (19') long by 4.88 m (16') wide by 1.83 m (6') to 2.13 m (7') high, cut as an extension of 2 South at 5 West.

OBJECTIVES AND SUMMARY

The objective of this contract was to measure the bearing pressure of floor strata at the base of the Galt Coal seam in a manner similar to the action of hydraulic chock roof supports.

A summary of the results of the seven tests is given in Table 1 below.

TABLE 1

Short term Tests 1st load cycle	Material	Max. Bearing pressure exerted kpa	psi	Penetration of floor mm	inches
Nos. 1 & 2	floor coal dry	6900+ to 9650+	1000+ to 1400+	4 to 10	0.17 to 0.38
No. 6	floor coal wetted	6900+	1000+	7	0.30
Nos. 3 & 4	floor coal dry	6800+ to 7600+	985+ to 1100+	8 to 15	0.32 to 0.61
No. 5	floor shale on which water was poured	7930+	1150+	9	0.37
Long Term Test					
No. 4a	Floor shale (15 hour test)	7600+ to 3000+	1100+ to 435+	43	1.705
Short Term Tests - Reload Cycle					
No. 2	Coal dry	9650+	1400+	10	0.38
No. 3	Shale dry	6800+	985+	28	1.1"

In test 2 in coal, the 1st cycle of loading gave a deformation of 4 mm (0.17") and a further 5 m (0.21") in the reload cycle to 6900 kpa (1000 psi), for a total penetration of 10 mm (0.38").

In test 3 in shale the 1st cycle of loading at the 4800 kpa (700 psi) level gave a deformation of 14 mm (0.57") and 4 mm (0.17") in the reload cycle. At the 6900 kpa (1000 psi) level this can be projected as 22 mm (0.85") to 6 mm (0.25") for a total penetration of

28 mm (1.1"). The tests were designed to represent, as near as is practical, the bearing pressure of longwall chock bases on the floor strata at the base of the Galt seam. In the short term tests, a systematic build-up of pressure against time allowed the upper limit of pressure to be reached and held over a significant period of time (15 minutes).

An interpretation of the above results, in the opinion of the CMRC engineers, is that the materials as tested show the floor coal and dry floor strata will bear a load in excess of the yield of hydraulic supports of a longwall face without failure. The tests also indicate that the bottom plates of the chocks may sink into the shale floor if left over a period of several days. During the period of the tests no wet conditions were available for further testing and CMRC was informed that material samples taken from the test site showed that the wetting of the shale did not affect the moisture content as tested in the samples taken at the test site.

TEST PROCEDURES

All test sites were levelled with an axe and all loose material removed prior to setting up the plate load equipment.

An initial small load of 690 kpa to 1035 kpa (100 to 150 psi) on the base was made to crush any minor debris that may have moved with the bottom plate while sliding it over to the level floor site. Later in the calculations the zero penetration point is derived for zero psi loading by extending the initial load portion of the pressure-penetration line to zero pressure on the pressure axis. See Charts, Appendix B. This additional minor amount of penetration is added to each point on the charts to give the complete pressure-penetration function and total penetration for the condition of no debris under the bottom plate. One test site each of coal and shale was reloaded.

The roof volume above the top plate was packed with timber. The plate load test equipment was loaded in increments of 345 kpa (50 psi) every 1 minute; level readings on the base were taken immediately on reading the desired pressure. Any upward or downward movement of the base was noted during the relaxation period after reaching the desired pressure. Note - during this test, the base moved no more than $\pm$0.01" between readings.

The loading rate and the number of readings taken adhere to 1981 ASTM, specifications D2664 and D3148 for determining the compressive strength of rock cores and their elastic moduli. These specifications state that for these strain rates strength values are reasonably free of rapid loading effects. ASTM has no specifications for plate load tests on rock.

DESCRIPTION OF TEST EQUIPMENT

The plate load test equipment consists of

(a) Enerpac 100 ton capacity hydraulic cylinder RR 10013

(b) Enerpac double acting pump - P464

(c) 8" nominal dia., ½" wall, 50,000 lb. yield (CSA G40. 21-50W) structural steel column in sections of 6', 4', 3', 2', 1'. Sections bolt together by means of flanges to form a unit operable at heights from 3' to 21'. Design capacity of column is 100 tons at 20'. The unit is complete with stabilizers to hold unit in place and a 3' long rod marked off in 0.01". The unit comes with a 12" x 12" square 1" thick steel bottom plate and a 12" x 18" top plate. Wooden blocking is used to increase roof bearing area to approximately 10 ft^2. Alignment of sections is to ±0.01". Faces of the column are machined right angles to lengths to match the flanges and are complete with alignment pins on the faces of flanges. Alternate base plates of 8" x 8" and 6" x 6" are available.

(d) Miscellaneous - 10,000 psi hydraulic hoses, fittings, 6" pressure guage to ½%, steel storage box, operating manuals, 3/4" unc bolts and nuts (CSA A325 or better), welding rod used - 7018 (70 ksi) spec. - CSA W591. All components are steel.

NOTE - Appendix E provides pictures of Plate Load Test Equipment.

PLATE SIZE - DETERMINATION 0.3 m x 0.3 m (12" x 12")

The USBM ran numerous tests using various sizes of plates in hard coal in longwall mines with a 100 ton ram. They found that the average bearing capacity of a coal seam tends to become constant when tests are made with rectangular (but not circular) plates of large area. They ran a series of 80 tests in 9 mines using plates of 0.15 m x 0.15 m (6" x 6") to 0.3 m x 0.3 m (12" x 12") and determined that the bearing capacity does not vary significantly with the size of the bearing plate within this range. They also state that the results of plate load tests may be used without any need for extrapolation. See Appendix F for copy of reference: In Situ tests at Bearing Capacity of Roof and Floor on selected Bituminous Coal Mines USBM R1-7406.

Numerous other tests have been done in the U.S. and by the NCB in the U.K. with 0.3 m x 0.3 m (12" x 12") plates. Dames and Moore have used a 100 ton Ram in the U.S. see Appendix F.

These tests indicate that a 0.3 m x 0.3 m (12" x 12") plate gives a very good approximation of the loading characteristics of the bases on the floor for longwall supports. These support bases are commonly some 0.5 m (20") wide.

Plate load tests are used to design hydraulic supports for longwall in the United States. See Appendix F, reference Design Optimization in Underground Coal Systems, Vol. 10.

TEST SITE CONDITIONS

The test site was at the intersection of 2 South and 5 West in the 5.8 m (19') long stub extension of 2 South; see Appendix D Plan: Test Sites. Nine tests were originally intended for the site, 3 on the coal floor and 3 on the shale floor and 3 in

the shale; see Appendix D: Sketches of Plate Load Test Site. The test site was cut with a section of shale floor 63 mm (2½") to 114 mm (4½") below the top of the shale and a section of floor coal 0.225 m to 0.3 m (9" to 12") thick. In co-operation with Dr. Bielenstein of AMCL, it was determined that the section of floor shale exposed was the weakest in the 0.7 metre shale horizon below the coal. Three shale tests were removed from the test program. An overnight test in wetted floor shale was incorporated into the test program.

The roof strata exposed was uneven at the test site and extended 0.9 m (3') above the top of the Galt coal seam; see Section - 2 South. It was well supported with steel sets and soft wood lagging. The test sites were chosen according to the floor conditions for evenness and for a maximum variation of thickness of shale or coal. Difficulty was experienced with the roof because the plate load test equipment could not load on the roof directly under the steel sets without additional timber packing. The lagging between the steel sets had to be repacked. An additional "I" beam was set close to the face and two "I" beams were cut 1.5 m (5') long so that the loading could bear on two steel sets on the roof at one time. With these modifications to the site, the site was found to be satisfactory for the plate load tests.

Before testing the floor under wet conditions, fifteen hours (or two shifts) of constantly wetting down the floor was performed by a Precambrian technician while taking pressure and level readings on an overnight pressure test. This later test started dry but was wetted down as well during both shifts. CMRC was later informed by AMCL that samples of floor shale taken both before and after wetting showed no significant change in moisture content. Total moisture in both cases was in the range of 9 to 10% on the basis of this information. The floor shale cannot be considered wet for the assessment of floor bearing pressure.

CALCULATIONS

Chart scales of pressure with range zero to 6900 kpa (1000 psi) or zero to 9650 kpa (1400 psi) and penetration with range zero to 89 mm (3.5") were chosen to demonstrate the range of both hard and soft coals if found. Soft coals may yield in the neighbourhood of 690 to 1580 kpa (100 to 200 psi), have penetrations of 76 to 114 m (3 to 4.5 inches), and a modulus of deformation of 690 to 2000 kpa/2.5 mm (100 to 300 psi/inch). Tests 2 and 3 are replotted with a second penetration scale to show the effect of reloading.

TEST RESULTS

The tests generally show an initial flattening of the shale or coal base and compression of floor cracks and separations of 2.5 mm to 10 mm (0.1"to 0.4") during the 1st 300 to 500 psi of loading. This is evidenced by a lower modulus of deformation (slope) during this period - see Charts. This is followed by a generally straight pressure -penetration line to the maximum pressure tested. Test (5) is an exception where the yield point of the wetted shale is apparently reached linearly at 7230 kpa (1050 psi). This is demonstrated by a significant reduction in the modulus after the yield point.

(2). COAL FLOOR TESTS - Test sites 1 and 2 dry, site 6 wetted.

The 0.225 m to 0.3 m (9" to 12") of coal tested was found to be hard with a strength in excess of 6900 kpa (1000 psi) under both wetted and dry conditions. The floor was prewetted for fifteen hours prior to testing the floor wet. After determining that the yield point of the coal was not reached at 6900 kpa (1000 psi), site 2 was reloaded to 9650 kpa (1400 psi), the limit of the plate load test equipment with the 0.3 m (12" square) foot plate. The coal withstood this pressure. Moreover, as the coal base was not cut perfectly flat, approximately 15% of the base plate was observed to be not in contact with the floor following the test. Therefore, the floor coal, when dry, withstood approximately 11,400 kpa or 1650 psi (1400/0.85).

Test site 2 was reloaded to 9650 kpa (1400 psi) after loading the first cycle to 1000 psi and subsequently retreating to the setting pressure of 1055 kpa (150 psi). Under a worst case sernario, the maximum plastic deformation is interpreted as 4.3 mm (0.17"), the deformation of the first loading cycle. Total deformation is 9.7 mm (0.38").

(3). SHALE FLOOR TESTS - Test Sites 3 and 4 dry, site 5 wetted.

The shale floor was tested as cut 64 mm to 114 mm (2.5" to 4.5") below the top of the shale. The shale was found to have a strength in excess of 6900 kpa (1000 psi) under both dry and wetted down conditions.

Test 3 was stopped at 4800 kpa (700 psi) as the two steel beams, mounted below the steel sets, buckled. Subsequently, the floor was reloaded to approximately 6900 kpa (1000 psi). Both tests 3 and 4 were stopped at 6900 kpa (1000 psi) due to major breakage of the wooden blocking above. The wet shale test 6 stopped at 7930 kpa (1150 psi) after an apparent yield point was reached at 7300 kpa (1050 psi) and for safety reasons when a steel set lifted off of its post at one end.

Test site 3 was reloaded to 6900 kpa (1000 psi) after loading the first cycle to 4800 kpa (700 psi) and subsequently retreating to the setting pressure of 1035 kpa (150 psi). Under a worst case senario, the maximum plastic deformation is interpreted as 21.6 mm (0.85"); see Chart - Test 3. As the reloading pressure reached 6900 kpa (1000+ psi) in a near- linear manner, it is interpreted that the first loading cycle would also have reached 6900 kpa (1000 psi) had the beams not buckled. Therefore, the total penetration reached on the reloading cycle is interpreted as being 28 mm (1.1"). This is well within the lifting height capacity of the bases of hydraulic roof support chocks.

(4). OVERNIGHT TEST - Test Site 4a in wet shale.

This test was the wetted down extension of test site 4. The pressure in the equipment was allowed to decrease naturally after an initial load of 7600 kpa (1100 psi). The pressure relaxed quickly to 5400 kpa (785 psi) after 12 minutes and then maintained a near-linear relationship of penetration (closure) with time of 40 mm (1.6")/day over the balance of the fifteen hour test; see Appendix C, Graph: Closure #4A. The site was kept wet by a Precambrian technician over this period.

After fifteen hours and a total penetration of 43 mm (1.7") for both tests 4 plus 4a, there was no evidence that the penetration would stop. Similarly on Chart Test 4a, penetration continued despite a great fall-off in pressure to 3000 kpa (435 psi). This indicates that penetration of the bottom plate into wet shale could be a problem on a longwall face. Pressures are still above the average borne by hydraulic face supports which are in the neighbourhood of 2000 to 3450 kpa (300 to 500 psi).

It is recommended that a 48 hour test be performed at the pressure levels expected to be borne by hydraulic face supports on a naturally wet shale floor. Total penetration in excess of 100 mm (4") could present a problem in moving chocks after standing still for a weekend. Such a test would show whether this problem could occur. CMRC has been informed that no naturally-wet shale floor conditions are available at the Kipp mine.

CONCLUSIONS

This report addresses the bearing pressure supportable by the floor strata below the Galt Coal Seam. The bearing pressure exerted by the plate load test equipment is similar to the action of hydraulic chock roof supports. Dry floor strata will bear a load in excess of the yield of hydraulic support of a longwall face without failure. It is apparent that wetted shale may present a problem of high penetration into the shale by the chock when it stands still over a period of days.

In our opinion the tests to date indicate that a continuous penetration would occur in wetted shale conditions, and in practice where normally wet or wetted shale is likely to be encountered, a thickness of floor coal would be left in the order of that in the test heading to maintain the operation of a longwall face.

D.H. Green, P. Eng.

P.L. Wright, P. Eng.

COAL MINING RESEARCH CENTRE

CONTRACT BETWEEN PRECAMBRIAN SHIELD RESOURCES LTD.
AND COAL MINING RESEARCH CENTRE
TO GOVERN PLATE LOAD BEARING TESTS AT KIPP MINE.

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1.0 SCOPE

This contract covers 8 plate load bearing tests at the Kipp Mine by the Coal Mining Research Centre for purposes of determining the bearing pressure of floor strata at the base of the Galt Coal Seam in a manner similar to the action of hydraulic chock roof supports.

2.0 CMRC AGREES TO

1) Provide supervision and operation of the equipment through provision of one Senior Project Engineer and one Junior Engineer on site chargeable to Precambrian Shield Resources Ltd. at standard rates (see Schedule of Rates -Section 6.0).

2) Provide surface transportation to the site from Edmonton chargeable at regular rates.

3) Provide a confidential report to Precambrian Shield Resources Ltd. on the work undertaken.

4) Co-operate with Precambrian Shield Resources Ltd. Engineers in the evaluation of all test data and in the choice of test sites.

5) Comply with Kipp Mine safety procedures and statutory management instructions while on mine property.

3.0 PRECAMBRIAN SHIELD RESOURCES LTD. AGREES TO

1) Provide transportation at the mine site of materials and equipment to and between test sites.

2) Provide the service of a shuttle car to support the test project throughout its duration.

3) Provide statutory supervision at the working place.

4) provide safety cables and anchoring points for stabilizing the test equipment as required.

5) Release all date from the tests to CMRC for compilation of a Confidential Report.

6) Pay for the services of a CMRC Senior Project Engineer and Junior Engineer at standard rates (see Schedule of Rates) on, to and from the mine site for the direction of the project and for food, accommodation and transportation of same also at standard rates (see Section 6).

7) Supply test sites underground in accordance with agreed to design and location in Section: Test Site Location and Design.

8) Provide timber to wedge the plate load test equipment: this to consist of fir and a minimum of 4" x 6" sq. x 24" timber, 1" and 2" thick boards and

wedges.

9) Provide underground surveying level or transit c/w tripod and a rod which reads to within 0.01 in. or 1 mm.

4.0 TEST SITE LOCATION AND DESIGN

Test sites are required according to the following specifications: see drawings nos. attached.

5.0 SCHEDULE OF WORK

Item of work and mandays required:

1 day to haul equipment
3 days - tests
1 day travelling time Edmonton to Lethbridge
1 day report.

6.0 OPERATING AND REPORT COSTS OF TEST

CMRC STAFF

Division Manager	1 day @ $550	$ 550.00
Senior Project Engineer	6 days @ $485	2,910.00
Junior Engineer	5 days @ $260	1,300.00
Meals (Lethbridge)	10 days @ $35	350.00
Hotel	10 days @ $40	400.00
Sub Total Staff charges.		$5,510.00

TRAVEL

Truck rental (including insurance)	8 days @ $40	$ 320.00
Km charge - Edmonton to Lethbridge_ 2 x 520 = 1040 @ 14¢		145.00
Lethbridge to Kipp 5 x 2 x 16 = 160 @ 14¢		23.00
Gas and Oil 1200 km @ 8.5¢		102.00
TOTAL ESTIMATE FOR 8 TESTS		$ 6,100.00

Additional tests, if not requiring further mobilization

2 tests per day	Senior Engineer	$ 485.00
	Junior Engineer	260.00
	Subsistance & Transport	200.00
	Provision for additional travel	55.00
Estimate for 2 additional tests		$ 1,000.00

The above charges are estimates and which will not be exceeded provided the project stays within the specified scope.

7.0 OPERATING PROCEDURES

1) The erection of the plate load test equipment shall be made with two CMRC employees plus shuttle car and operator.

2) The erected extension cylinder shall be secured to posts or rock bolts with cable.

3) The operation, of the hydraulics and the reading of the instruments (Pressure guages and surveying instrument) shall be by CMRC employees.

4) The loading of the floor by means of the plate load test equipment shall be according to the following procedures:

a) Assemble all equipment in place on test site, load to 100 psi to test for leaking hydraulic lines.

b) Preload floor to 300 psi on 12" x 12" bottom plate (not guage reading). Increase load on base in increments of 50 psi every two minutes until failure or until 1000 psi loading is reached (See Chart: Mechanism of Loading). Measure deflection in floor with surveying instrument at mid point in two minute cycle. Record deflection and pressure readings. Failure is defined as that pressure which loads the floor to its ultimate strength (See Chart: Typical Stress Strain Curve), or a deflection of 6".

c) Test sites ar e to be no closer than 5' apart. Floor is to be chiseled level to fit the 12" x 12" bottom plate.

8.0 TERMS

CMRC to bill Precambrian Shield Resources lt d. of the agreed rates on completion of contract.

SIGNING AUTHORITY FOR PRECAMBRIAN SHIELD RESOURCES LTD.	SIGNING AUTHORITY FOR COAL MINING RESEARCH CENTRE
_____	_____
TITLE V.P. - Corp.	TITLE Managing Director.
DATE 4 / OCTOBER / 1982	DATE 5/10/82
_____	_____
TITLE _____	TITLE _____
DATE _____	DATE _____

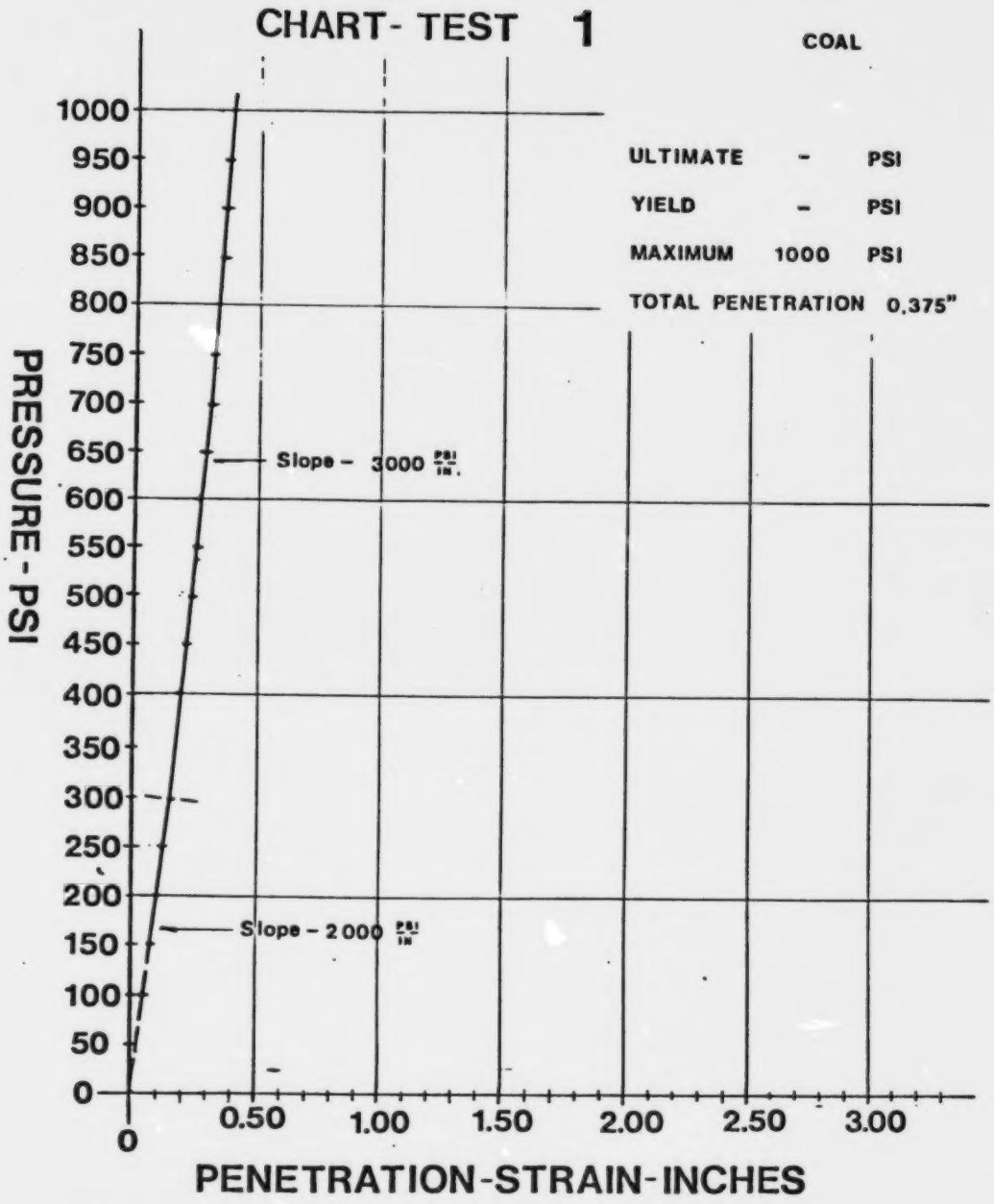


CHART- TEST 1
COAL
ULTIMATE - PSI
YIELD - PSI
MAXIMUM 1000 PSI
TOTAL PENETRATION 0.375"
Slope - 3000 PSI/IN.
Slope - 2000 PSI/IN
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

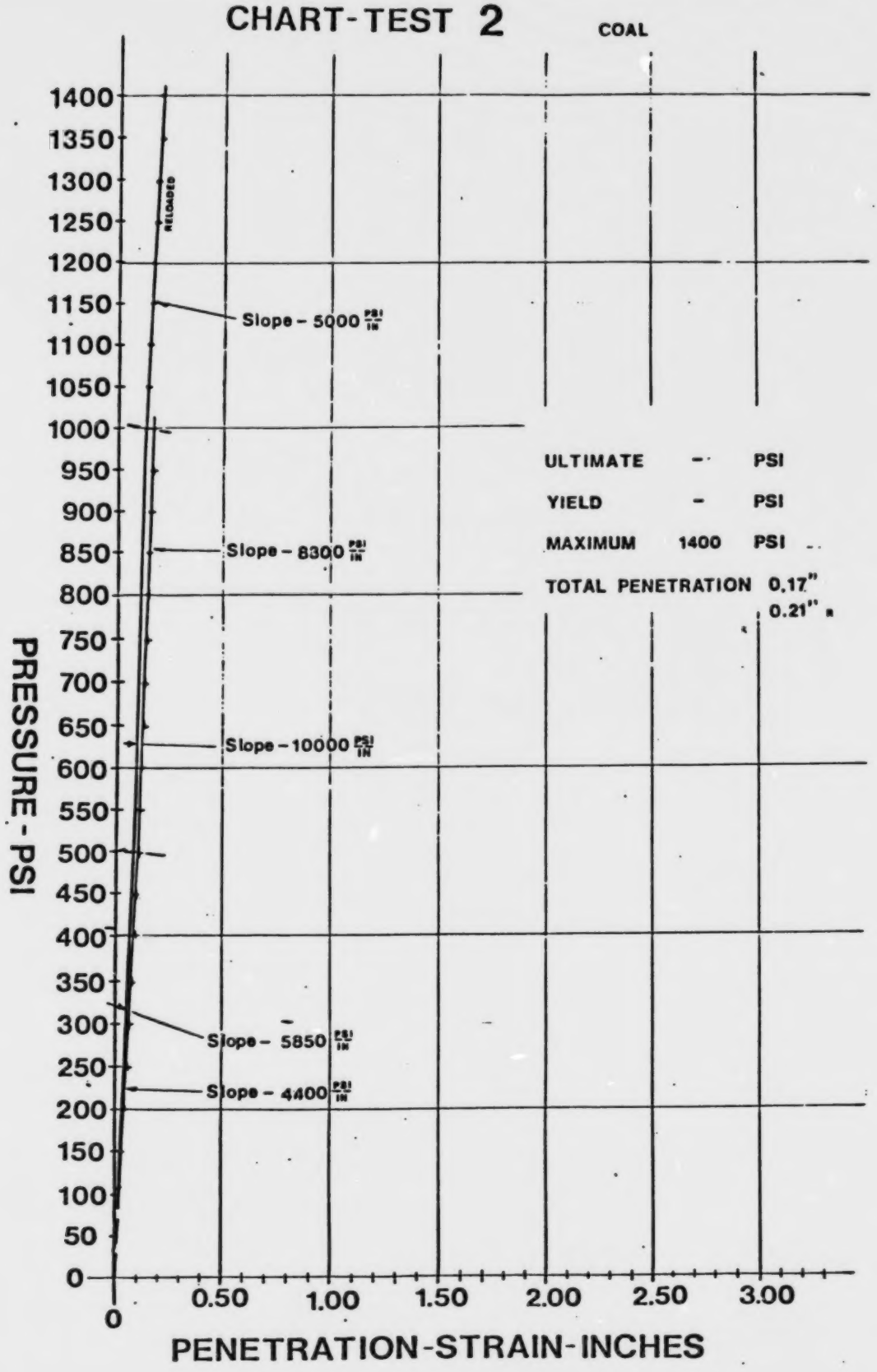


CHART-TEST 2
COAL
1400
1350
1300
1250
1200
1150
1100
1050
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
RELOADED
Slope – 5000 PSI/IN
Slope – 8300 PSI/IN
Slope – 10000 PSI/IN
Slope – 5850 PSI/IN
Slope – 4400 PSI/IN
ULTIMATE – PSI
YIELD – PSI
MAXIMUM 1400 PSI
TOTAL PENETRATION 0.17"
0.21"
PRESSURE - PSI
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

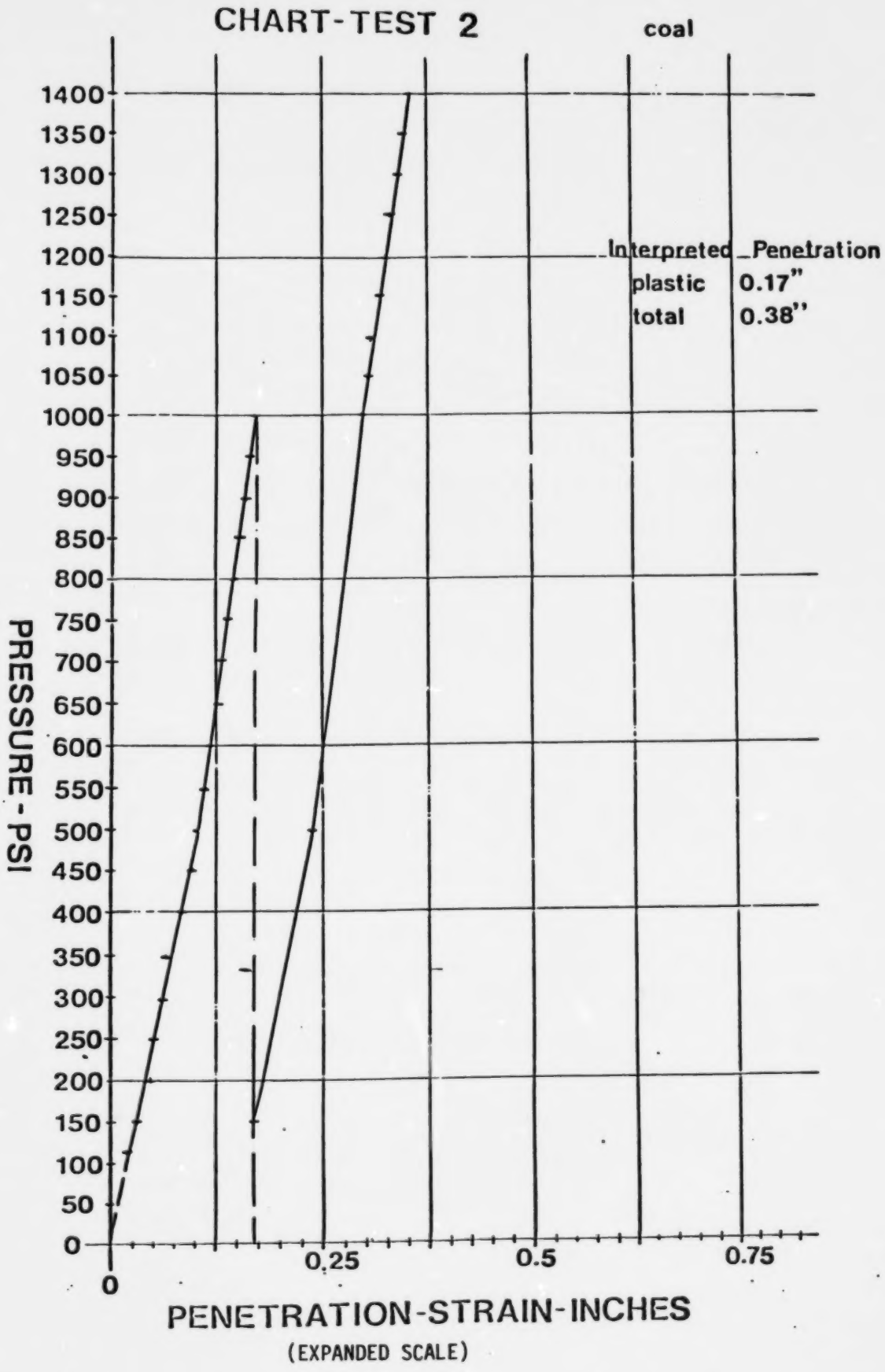


CHART-TEST 2
coal
Interpreted Penetration
plastic 0.17"
total 0.38"
PRESSURE-PSI
1400
1350
1300
1250
1200
1150
1100
1050
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.25
0.5
0.75
PENETRATION-STRAIN-INCHES
(EXPANDED SCALE)

CHART- TEST 3

SHALE

ULTIMATE - PSI

YIELD - PSI

MAXIMUM 988 PSI

TOTAL PENETRATION 0.57"

0.32" e

Slope - 2450 PSI/IN

Slope - 1250 PSI/IN

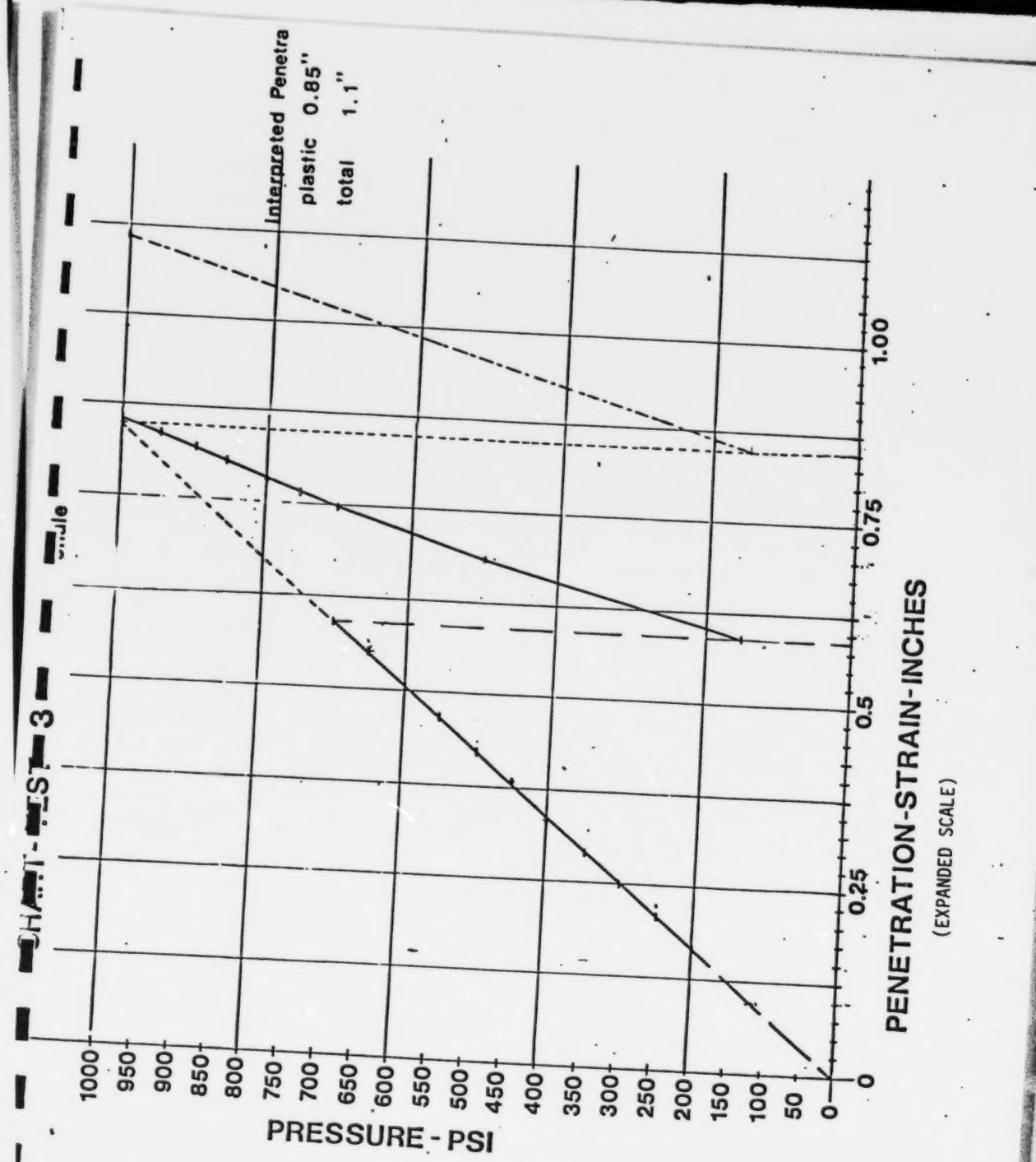
Slope - 2950 PSI/IN

PRESSURE - PSI

1000 950 900 850 800 750 700 650 600 550 500 450 400 350 300 250 200 150 100 50 0

0 0.50 1.00 1.50 2.00 2.50 3.00

PENETRATION-STRAIN-INCHES


Interpreted Penetra
plastic 0.85"
total 1.1"
CHART - TEST 3
PRESSURE - PSI
1000 950 900 850 800 750 700 650 600 550 500 450 400 350 300 250 200 150 100 50 0
0 0.25 0.5 0.75 1.00
PENETRATION-STRAIN-INCHES
(EXPANDED SCALE)

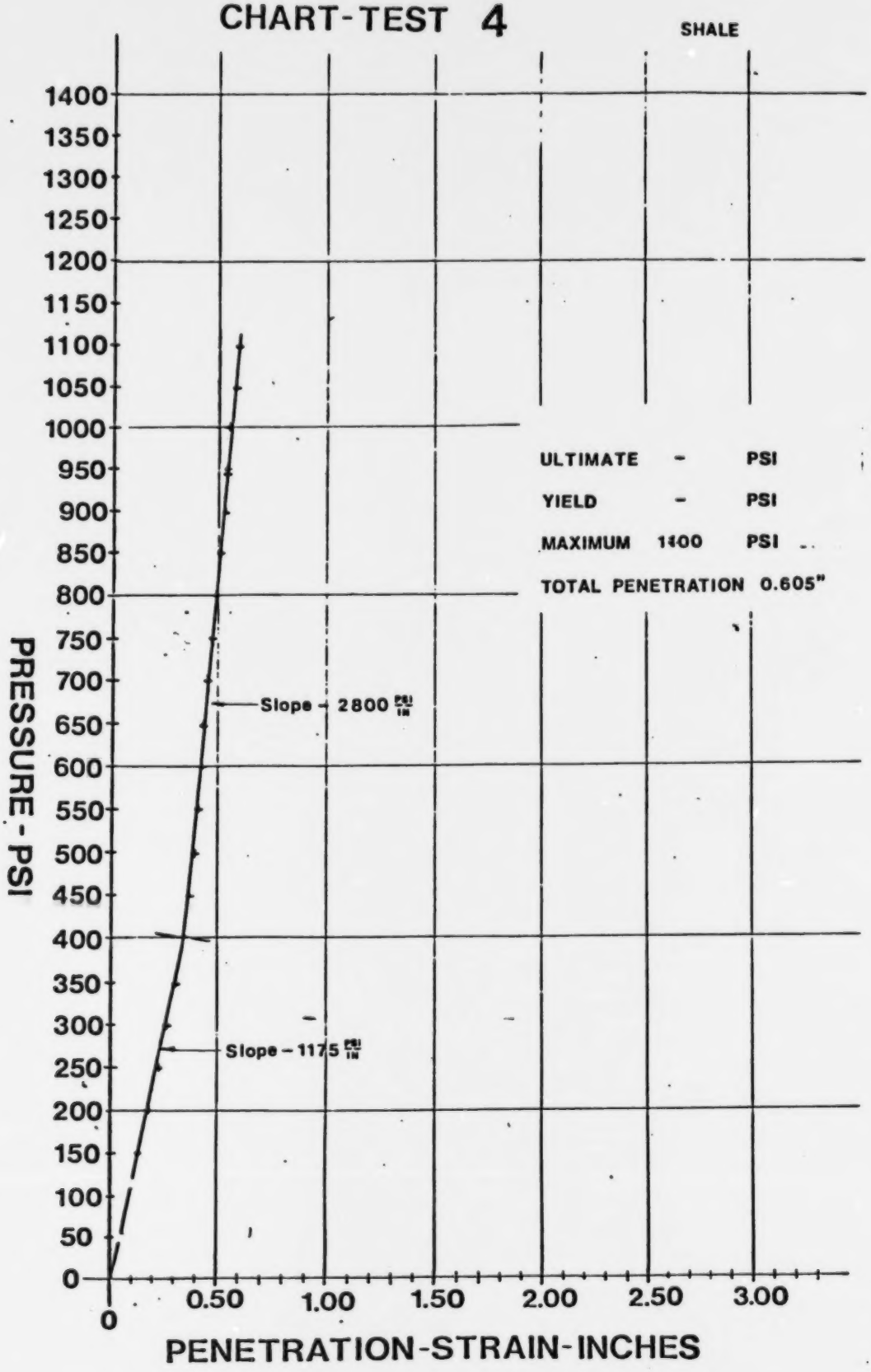


CHART-TEST 4
SHALE
1400
1350
1300
1250
1200
1150
1100
1050
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
ULTIMATE - PSI
YIELD - PSI
MAXIMUM 1400 PSI
TOTAL PENETRATION 0.605"
Slope = 2800 PSI/IN
Slope = 1175 PSI/IN
PRESSURE - PSI
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

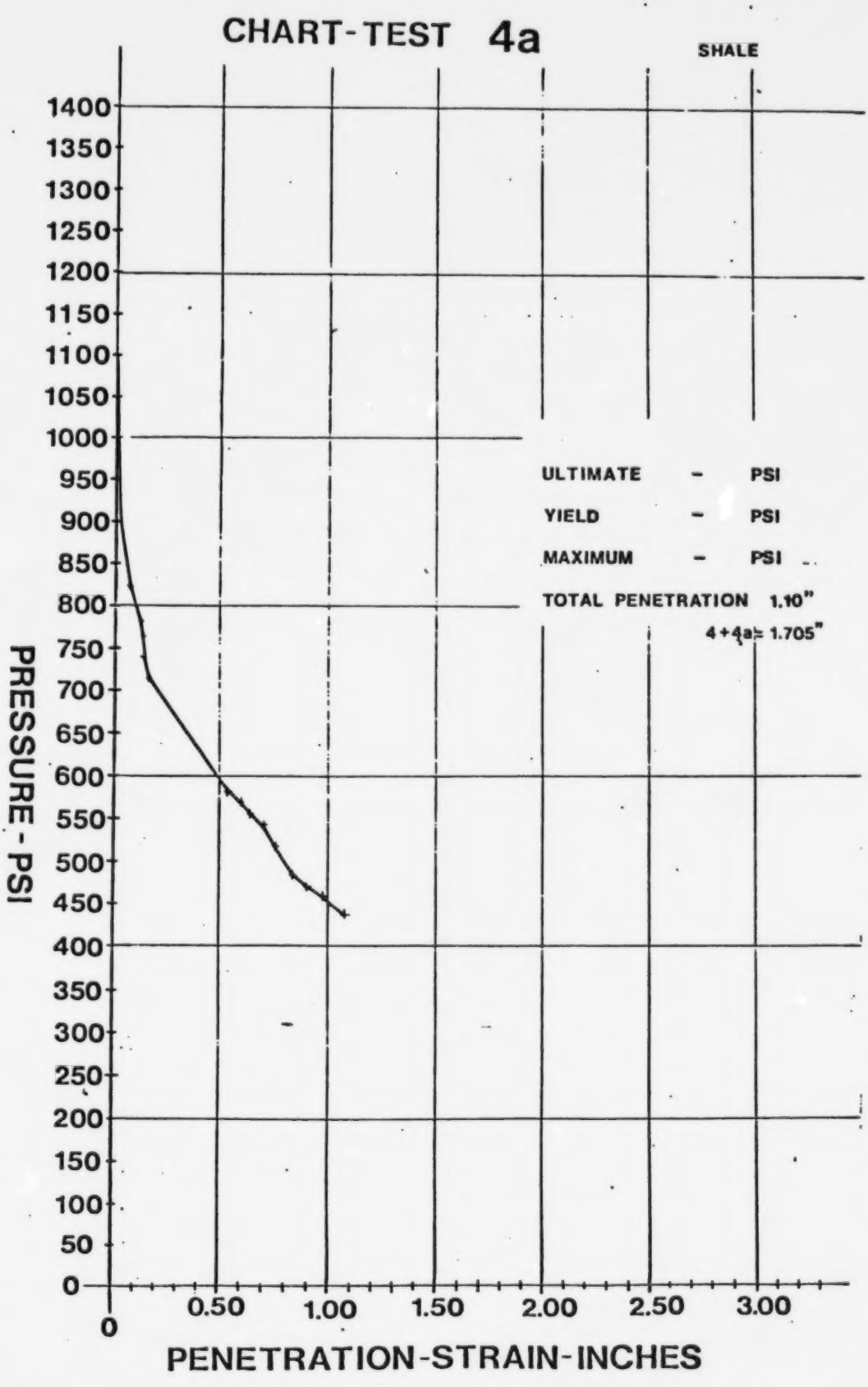


CHART-TEST 4a
SHALE
1400
1350
1300
1250
1200
1150
1100
1050
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
PRESSURE-PSI
ULTIMATE - PSI
YIELD - PSI
MAXIMUM - PSI
TOTAL PENETRATION 1.10"
4+4a= 1.705"
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

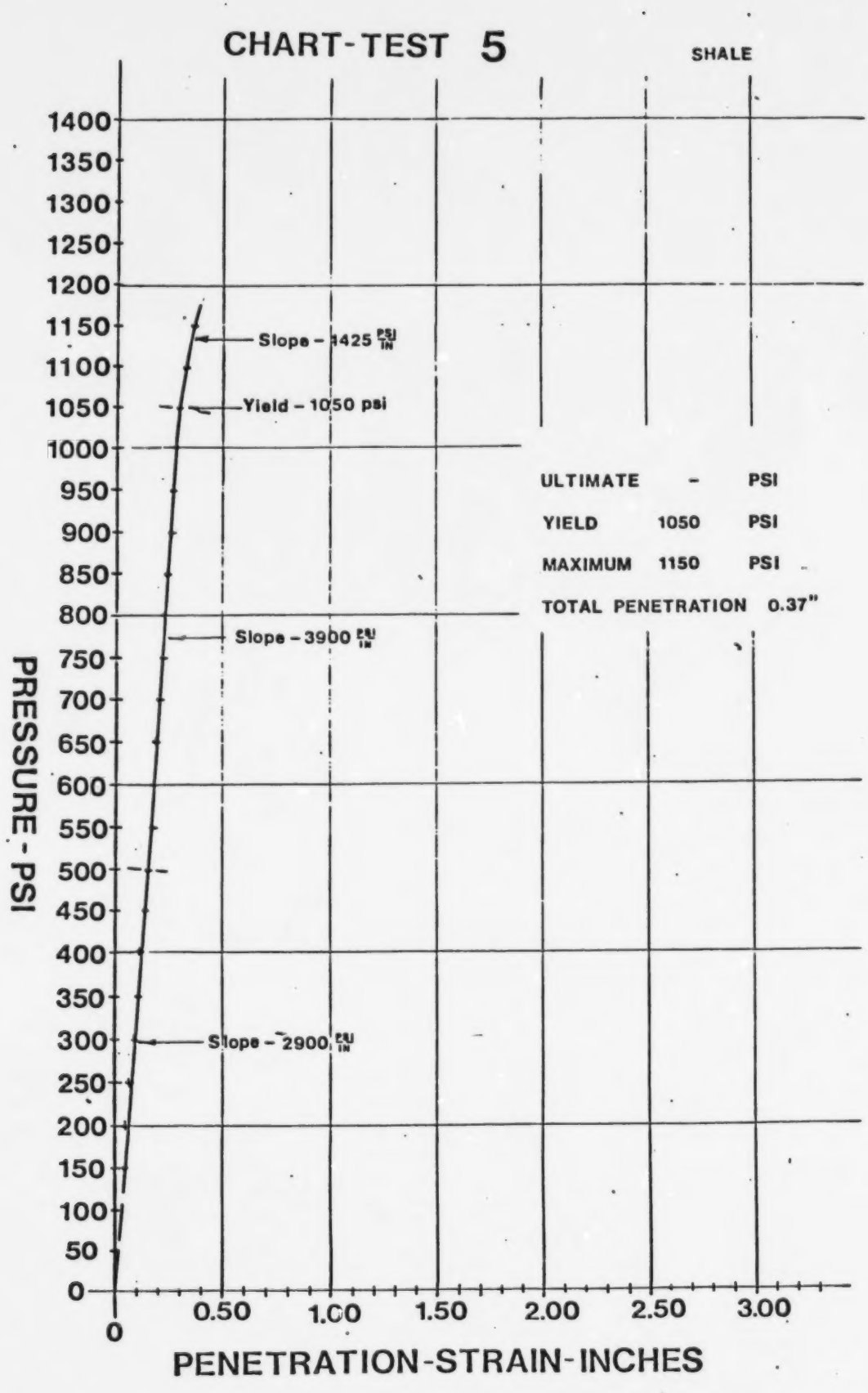


CHART-TEST 5
SHALE
Slope – 1425 PSI/IN
Yield – 1050 psi
Slope – 3900 PSI/IN
Slope – 2900 PSI/IN
ULTIMATE – PSI
YIELD 1050 PSI
MAXIMUM 1150 PSI
TOTAL PENETRATION 0.37"
PRESSURE - PSI
1400
1350
1300
1250
1200
1150
1100
1050
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

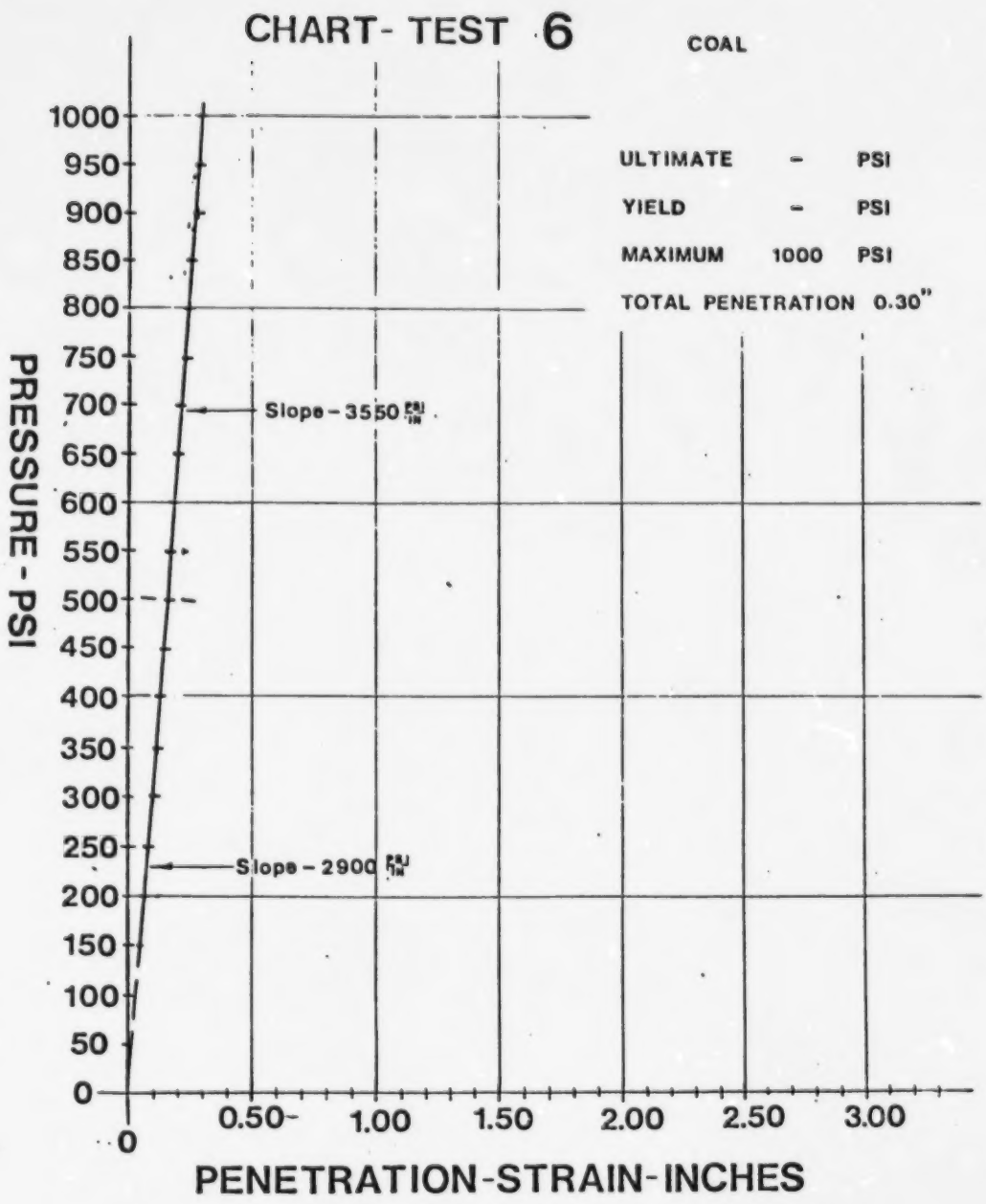
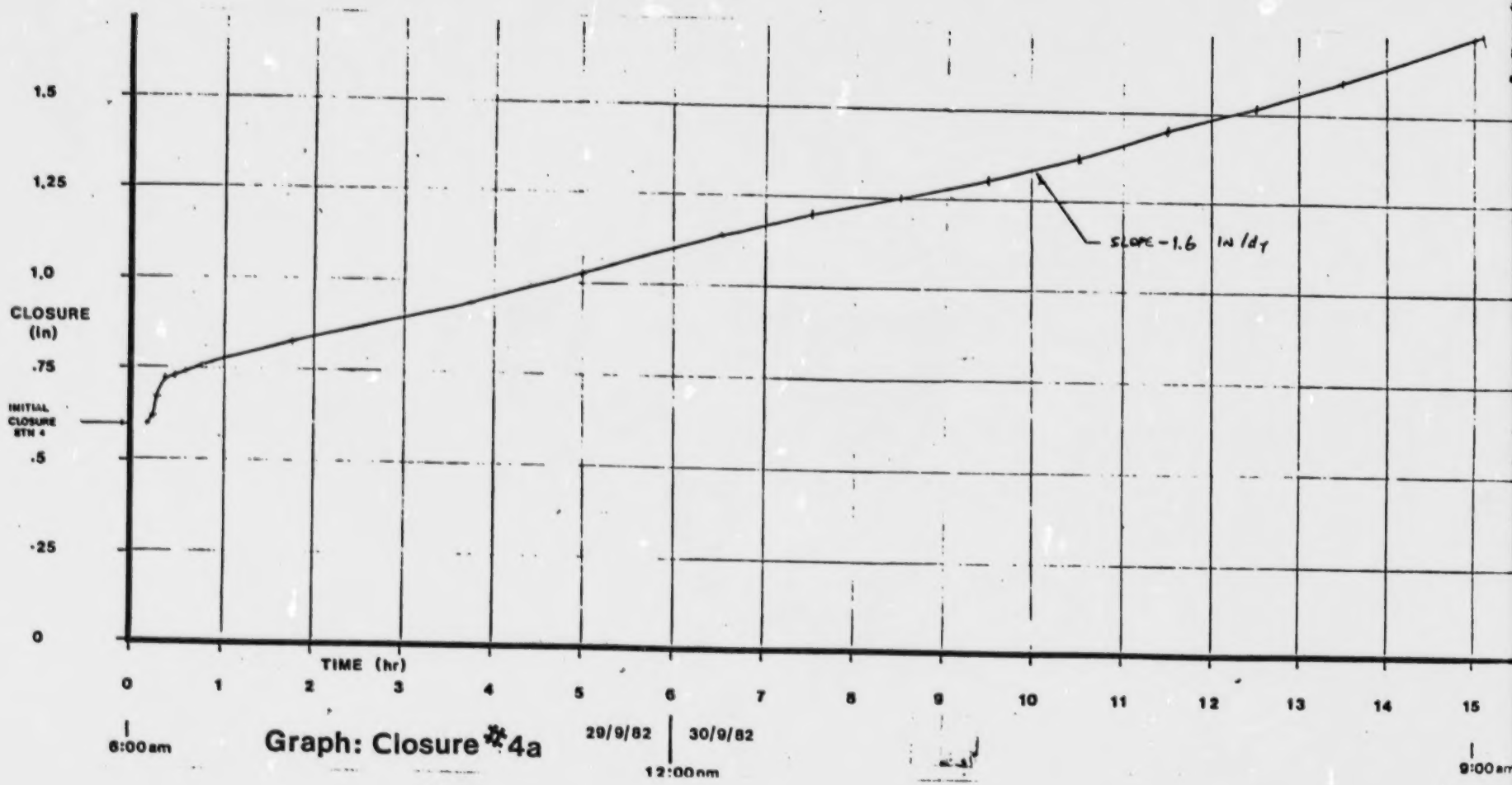


CHART- TEST 6
COAL
ULTIMATE - PSI
YIELD - PSI
MAXIMUM 1000 PSI
TOTAL PENETRATION 0.30"
Slope - 3550 PSI/IN
Slope - 2900 PSI/IN
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES



Graph: Closure *4a

CLOSURE STATION NO. — DATE/TIME OF TEST

CONDITIONS: WET

LOCATION 1 South 5 West Stub DATE MINED —

DATE	TIME	READ BY:	TRANSVERSE			VERTICAL								
						Stn 4 : ℄ of Entry.			Pressure - PSI and Place					
			READING	Closure	Cum Closure	READING	Closure	Cum Closure	~~READING~~	Closure	Cum Closure	READING	Closure	Cum Closure
29/9/82	6.10 PM	S.T. Cmrc				19.27	-	-	1100					
"	6:14	"				19.29	0.02	0.02	900					
"	6:17	Pre Camb.				19.34	0.05	0.07	825					
"	6:22	"				19.39	005	0.12	785					
"	6:28	"				19.40	0.01	0.13	765					
"	6:35	"				19.41	0.01	0.14	740					
"	6:45	"				19.43	0.02	0.16	715					
"	7.45	"				19.51	0.08	0.24	675					
"	9:45	"				19.61	0.10	0.34	625					
"	11:00	"				19.70	0.09	0.43	600					
30/9/82	12:30 AM	"				19.81	0.11	0.54	585					
	1:30	"				19.87	0.06	0.60	570					
	2:30	"				19.97	0.05	0.65	560					
	3:30	"				19.98	0.06	0.71	543					
	4:30	"				20.04	0.06	0.77	515					
	5.30	"				20.12	0.08	0.85	485					
	6:30	"				20.19	007	0.92	470					
	7:30	"				20.26	0.07	0.99	457					
	9.00	St. Cmrc				20.37	0.11	1.10	435					

PLATE LOAD TEST

CMRC

TEST NO.: (1) DATE: SEPT. 29/82

LOCATION: 2 South, 5 West - IN stub ENGINEER: D.H.G.

ON 12' coal OPERATOR: S.T.

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION IN	CUMULATIVE DEFLECTION IN	COMMENTS
1 min. between readings	700 (50)		100	18.785	(.004" to 200 psi)	
	1050			18.81	0.025	
	1400		200	18.84	0.055	
	1750			18.86	0.075	
	2100		301	18.885	0.10	
	2450		351	-		— added additional wedges & timber to compression pad on ROOF above plate load test COUPE
	2800		401	18.92	0.135	
	3150		451	18.945	0.165	
	3500		502	18.97	0.185	
	3850		552	18.985	0.20	
	4200		602	19.00	0.215	
	4550		652	19.02	0.235	
	4900		702	19.04	0.255	
2 min. between readings	5250		752	19.055	0.27	— Relaxation to 1800 psi in 30 secs (Timbers compressing and cracking)
	5600		802	19.07	0.285	
	5950		853	19.09	0.305	
	6300		903	19.10	0.315	
	6650		953	19.12	0.325	
	7000		1003	19.13	0.335	+0.04 = 0.375" Total Penetration

PLATE LOAD TEST

CMRC

TEST NO.: (2) DATE: Sept 29/82

LOCATION: 2 South, 5 West - IN stub ENGINEER: DHC

ON 10" coal OPERATOR: ST

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.83/144	LOAD psi	DEFLECTION IN	CUMULATIVE DEFLECTION IN	COMMENTS
1 min betw readings	~~700~~ 800		~~400~~ 115	18.19	– (10 x 1000 psi)	Deload: 1618 (1000 psi) Def. – C. Def. – (1000 psi) 0 IN
	1050			18.20	0.01	
	1400		200	18.22	0.03	
	1750			18.23	0.04	
	2100		301	18.24	0.05	
	2450		351	18.25	0.06	
	2800		401	18.26	0.07	
	3150		451	18.27	0.08	
	3500		502	18.28	0.09	16.24 0.06
	3850		552	18.28	0.09	
	4200		602	18.29	0.10	
	4550		652	18.30	0.11	Note: approx 85 lb of load on coal
	4900		702	18.30	0.11	
	5250		752	18.31	0.12	
	5600		802	18.32	0.13	
	5950		853	18.32	0.13	
	6300		903	18.33	0.14	
	6650		953	18.34	0.15	
	7000		1003	18.34	0.15	16.29 0.11
	7350			–	–	16.30 0.12
	7700			–	–	16.31 0.13
	8050			–	–	16.32 0.14
	8400			–	–	16.325 0.145
	8750			–	–	16.34 0.16
	9100			–	–	16.345 0.165
	9450			–	–	16.36 0.18
	9800		1400	–	–	16.37 0.19
	10.000		1425	–	–	– –

PLATE LOAD TEST

MRC

TEST NO.: (3) DATE: Sept 29/82

LOCATION: 2 South, 5 West - in stub ENGINEER: D.H.C.

on floor shale OPERATOR: S.T.

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION in	CUMULATIVE DEFLECTION in	COMMENTS
					(+.12" to 0")	Re Run Def. C Def. (+.03" to 0")
	700		100	-		
1 min. betw Readings	1000			23.76	-	19.67 -
	1400		200	23.81	0.05	
	1750			23.84	0.08	
	2100		301	23.88	0.12	
	2450		351	23.92	0.16	
	2800		401	23.96	0.20	
	3150		451	24.02	0.26	
	3500		502	24.05	0.29	19.77 0.10
	3850		552	24.09	0.33	
	4200		602	24.13	0.37	
	4550		652	24.18	0.42	
	4900		702	24.21	~~0.45~~	19.84 0.17 (2@6" I beams Crush. - Re Run Test
	5250		752	-		19.85 0.18
	5600		802	-		19.87 0.20
	5950		853	-		19.90 0.23
	6300		903	-		19.92 0.25
	6650		953	-		19.94 0.27
	~~7000~~ 6900		1003 985	-		19.96 0.29
						STOP: Timber Roof Crushing.

PLATE LOAD TEST

CMRC

TEST NO.: (4)

DATE: Sept 29/82

LOCATION: 2 South, 5 West -1N stub on floor shale

ENGINEER: D.H.G.

OPERATOR: S.T.

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION	CUMULATIVE DEFLECTION	COMMENTS
(min. before Readings	700		100	–	(1.13" to – 0ni)	① Overnight test was Run after watering Floor on shale - see Test H (9a)
	1000			18.795	–	
	1400		200	18.85	0.055	
	1750			18.90	0.105	
	2100		301	18.94	0.145	
	2450		351	18.98	0.185	
	2800		401	19.02	0.225	
	3150		451	19.04	0.245	
	3500		502	19.06	0.265	
	3850		552	19.08	0.285	
	4200		602	19.10	0.305	
	4550		652	19.11	0.315	
	4900		702	19.12	0.325	
	5250		752	19.14	0.345	
	5600		802	19.16	0.365	
	5950		853	19.18	0.385	
	6300		903	19.20	0.405	
	6600 to 6650		953	19.21	0.415	– observed that no change occurs in dial reading over 1 minute.
	7000		1003	19.22	0.425	
	7350			19.25	0.455	
6:10 PM	7700		1103	19.27	0.475	
6:14 PM	6200			① –	–	STOP: 2 persons to Pump hrd. Pump out at "energy".
						+0.13 : 0.605" total Penetration

PLATE LOAD TEST

MRC

TEST NO.: (4a) Overnight Test DATE: Sept 29-30/92

LOCATION: 2 South, 5 West - IN stub ENGINEER: DHG

on floor Shale OPERATOR: S.T.

	TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION	CUMULATIVE DEFLECTION	COMMENTS Relaxation Test:
29th	6:10	7300		1100	19.27	-	
	6:14 pm	6300		900	19.29	0.02	Flow watered as Pressure on shiner from test (4). Precambrian Technician kept sites (4), (5), (6) watered all night until 730 am shift change. Water ponded on shale sites, but ran off red site as it sloped down to shale
	:17	5750		825	19.34	0.07	
	:22	5500		785	19.39	0.12	
	:28	5350		765	19.40	0.18	
	:35	5200		740	19.41	0.18	
	6:45	5000		715	19.43	0.16	
	7:45	4700		675	19.51	0.24	
	9:45	4400		625	19.61	0.34	
	11:00	4200		600	19.70	0.43	
30th	0:30	4100		585	19.81	0.54	
	1:30	4000		570	19.87	0.60	
	2:30	3900		555	19.92	0.65	
	3:30	3800		543	19.98	0.71	
	4:30	3600		515	20.04	0.77	
	5:30	3400		485	20.12	0.85	
	6:30	3300		470	20.19	0.92	
	7:30 am	3200		457	20.26	0.99	
	9.00 am	3050		435	20.32	1.10	29th 6.00 pm to 30th 9:00 am: 15 Hr Relaxation Test

PLATE LOAD TEST

CMRC

TEST NO.: (5) DATE: Sept 30/82

LOCATION: 2 South. 5 West -1m stub on floor shale (wet) ENGINEER: D.H.C

OPERATOR: S.T

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION	CUMULATIVE DEFLECTION	COMMENTS
	700		100	—	(+0.01" to 0.01")	Wetted floor 13.5 HRS continuously prior to loading test site (5,6) (wet both 7:30 am to 9:00 am) Floor damp on arrival 9:00 AM.
1 min between readings	1000			18.80	—	
	1400		200	18.81	0.01	
	1750			18.83	0.03	
	2100		301	18.85	0.05	
	2450		351	18.87	0.07	
	2800		401	18.88	0.08	
	3150		451	18.90	0.10	
	3500		502	18.92	0.12	
	3850		552	18.93	0.13	
	4200		602	18.94	0.14	
	4550		652	18.95	0.15	
	4900		702	18.97	0.17	
	5250		752	18.98	0.18	Relaxation 1' (20 sec) — 4850 (down 0.01")
	5600		802	18.99	0.19	— 5150
	5950		853	19.00	0.20	— 5550
	6300		903	19.02	0.22	— 5820 (155 sec)
	6650		953	19.03	0.23	— 6200
	7000		1003	19.04	0.24	— 6350 (7350 = 1m)
	7350			19.06	0.26	— beam lifted 2" off Post on W. side entry.
	7700			19.09	0.29	— reset to 7700: 19.10 (Yielding!)
	8050		1154	19.13	0.33	+0.04" = 0.37" Total Penetration

PLATE LOAD TEST

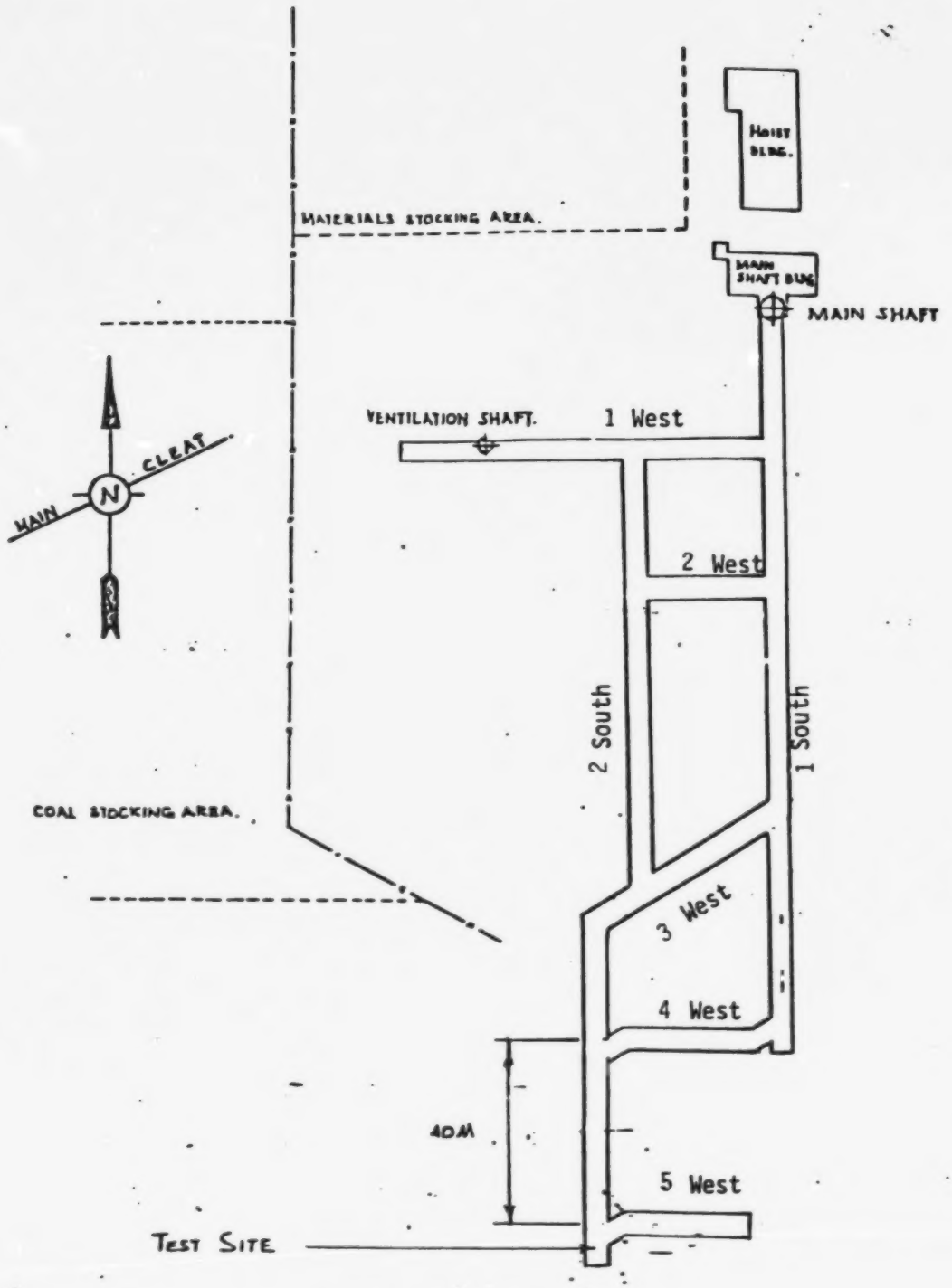
UMRC

TEST NO.: (2) DATE: Sept 30/82

LOCATION: 2 South, 5 West - in stub ENGINEER: D.H.G.

on 9" coal (wet) OPERATOR: S.T

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.63/144	LOAD psi	DEFLECTION	CUMULATIVE DEFLECTION	COMMENTS
	700		100	-	(700 to 700 = 0 psi)	Watered Floor 13.5 hrs Continuously PRIOR to loading test sites (S16 Floor damp during test
1 min btw Readings	1000			17.16	-	
	1400		200	17.19	0.03	
				17.20	0.04	
	2100		301	17.22	0.06	
	2450		351	17.24	0.08	
	2800		401	17.25	0.09	
	3150		451	17.26	0.10	
	3500		502	17.28	0.12	
	3850		552	17.29	0.13	
	4200		602	17.30	0.14	
	4550		652	17.32	0.16	
	4900		702	17.33	0.17	
2 min btw. Readings	5250		752	17.35	0.19	
	5600		802	17.37	0.21	
	5950		853	17.38	0.22	
	6300		903	17.40	0.24	
	6650		953	17.41	0.25	Stopped - 1000 psi - excessive compression on load cell
	7000		1003	17.42	0.26	Deflection: 6100 psi of the 30 secs
						0.26 + 0.04 = 0.30" Total Deflection



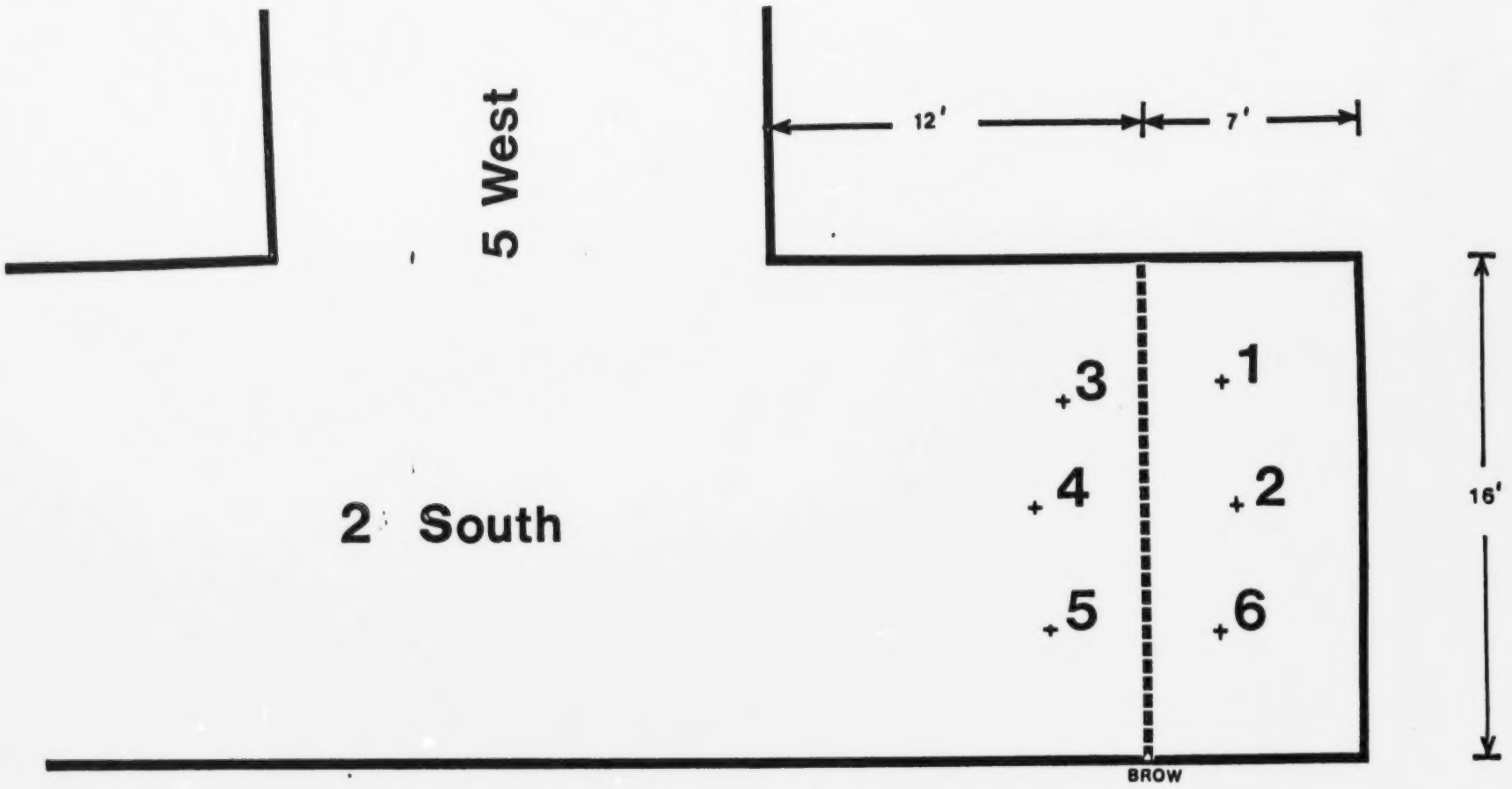
KIPP MINE - GALT SEAM

PLAN SHOWING THE POSITION OF THE CMRC TEST SITE

SCALE :- 0 10 20 30 40 metres

PLAN: TEST SITES

CMRC

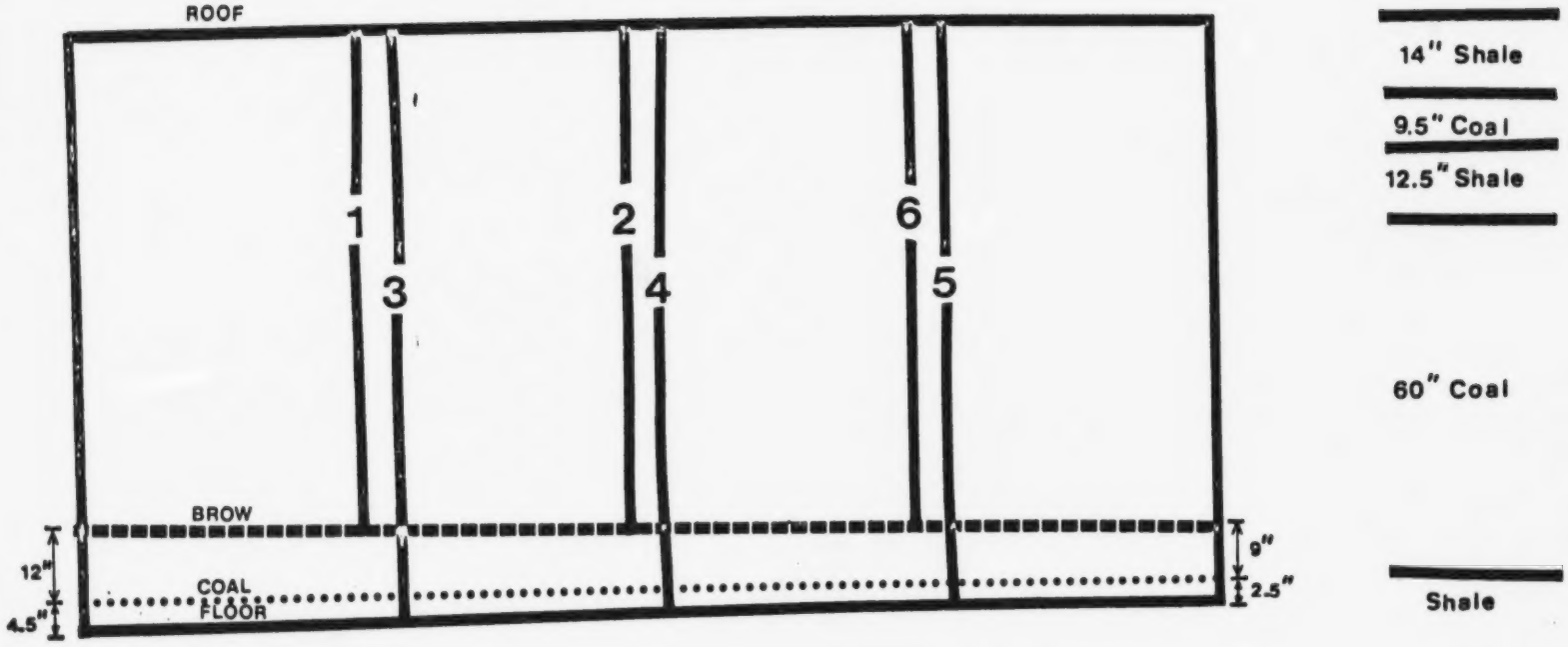


Precambrian Shield Resources Ltd
Kipp Mine

scale : 1 in = 5 ft

October 1982

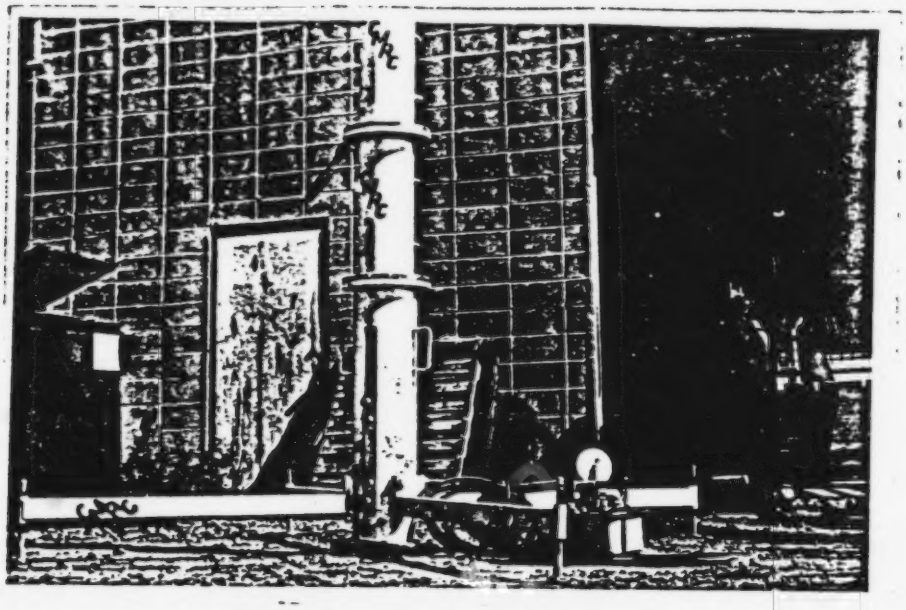
SECTION: 2 SOUTH



scale : 1 in = 30 in

Precambrian Shield Resources Ltd
Kipp Mine

October 1982



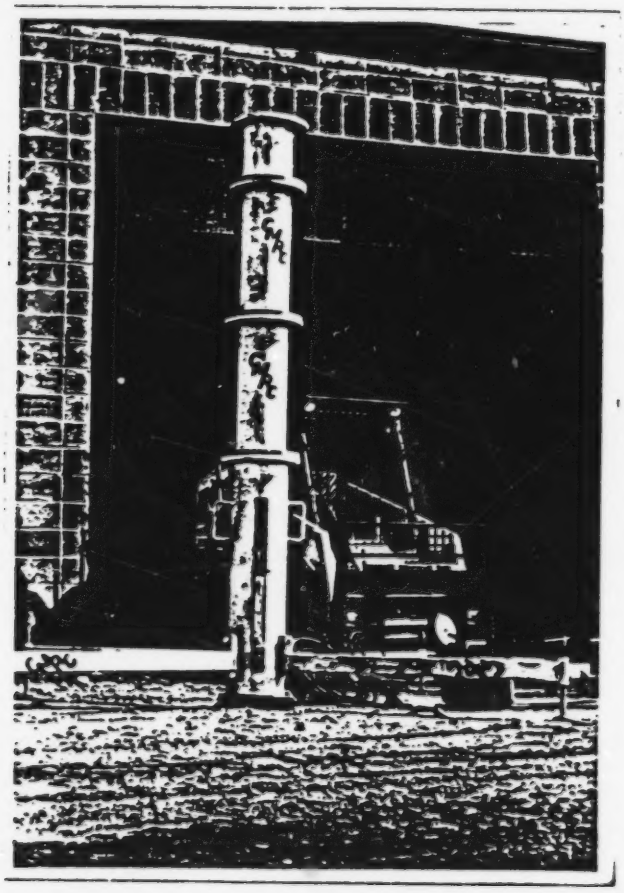


Plate Load Test Equipment

Dames & Moore Plate Load Test
- 100 ton USA - Old Bed Mine

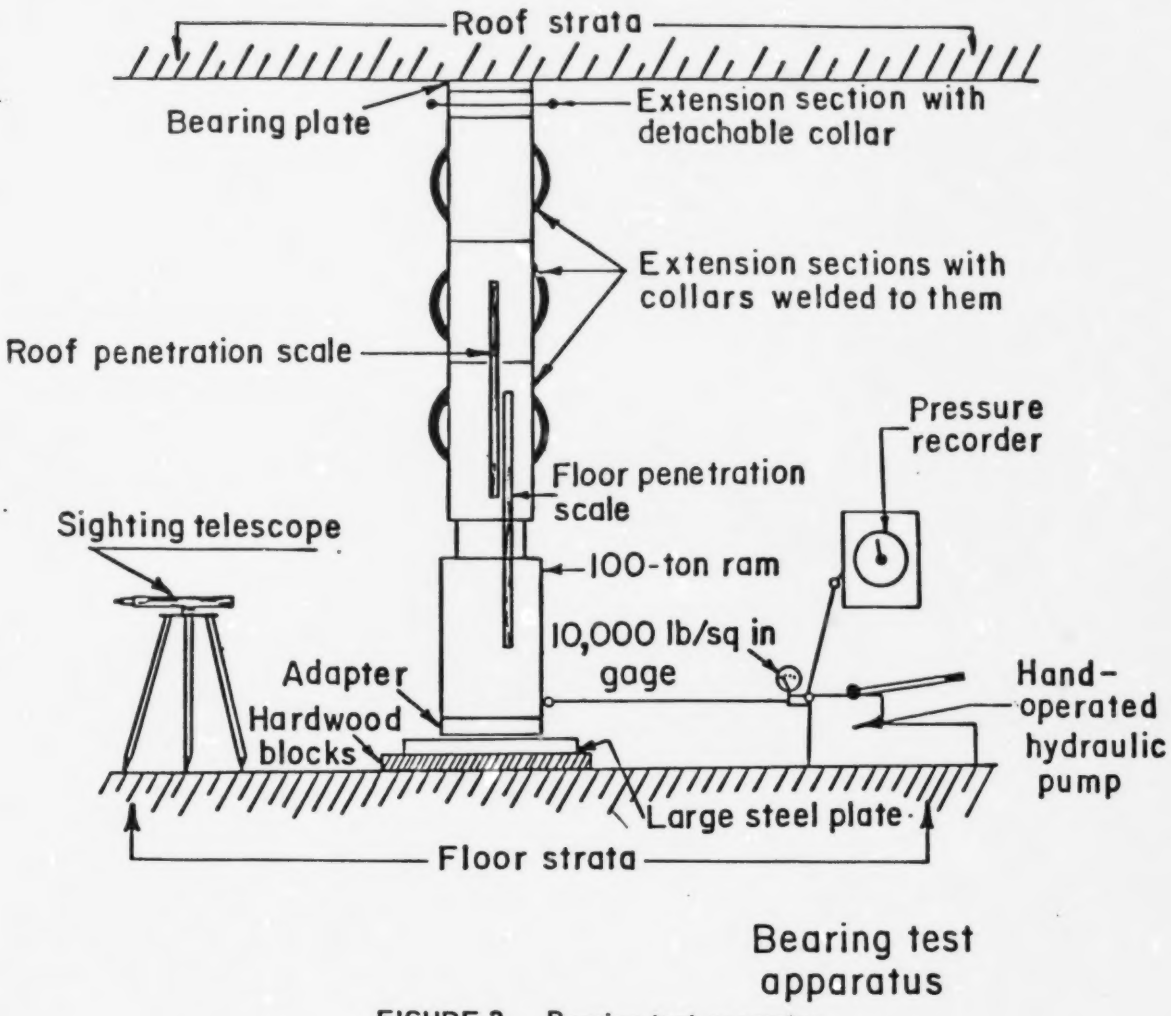


FIGURE 2. - Bearing test apparatus.

2. Geological study comprising rock-strata and faulting information.

3. Stress analysis on Rheinstahl chocks salvaged from a longwall face abandoned in 1967.

Dames and Moore concluded with the following:

1. A minimum of 6 inches of floor coal should remain intact. According to tests, the soft underclay fails at 300 lb/sq in when wet, whereas the floor coal can bear at least 1,000 lb/sq in. Therefore, the continuous mining machine in the panel development and the shearer at the face should cut to the roof rock and leave enough floor coal to maintain a mining height of 7 feet in the 8.5- to 9-foot-thick No. 6 coalbed.

US.

U.S.B.M. I.C. 8776. - Old Ben Mine 24 No 6 Coal, Ill

15.360

COAL MINING RESEARCH CENTRE

IN SITU TESTS OF BEARING CAPACITY OF ROOF AND FLOOR IN SELECTED BITUMINOUS COAL MINES

A Progress Report - Longwall Mining

By A. J. Barry and Oscar B. Nair

report of investigations 7406

UNITED STATES DEPARTMENT OF THE INTERIOR

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IN SITU TESTS OF BEARING CAPACITY OF ROOF AND FLOOR IN SELECTED BITUMINOUS COAL MINES

A Progress Report—Longwall Mining

by

A. J. Barry[1] and Oscar B. Nair[2]

ABSTRACT

A practical method has been developed and field tested which provides an estimate of the bearing capacity of mine roof and floor. The relationship between penetration of the strata and imposed loads is determined by in situ tests. Information of this type is needed to determine the maximum allowable pressure that may be applied by self-advancing supports on roof and floor of a longwall face without causing a punching type of failure.

INTRODUCTION

This study is part of a continuing effort by the Ground Control Technology Group of the Bureau of Mines to develop engineering information and to determine parameters that will be of use in the design of longwall mining systems.

With the advent of self-advancing hydraulic supports or chocks having design capacities of up to 600 tons, and the prospect that longwall mining will find increasing popularity in U.S. coal mines, there is a great need for engineering information that will help the operator to select his longwall support equipment and minimize the possibility of failure after installation underground.

Chocks of large yield loads currently being used in the United States can and do periodically apply large bearing pressures to the roof and floor strata at operating longwall faces. High bearing pressures can be reduced either by increasing the contact areas of the bearing plates or by reducing the yield load of the chocks below the design value. If the mine roof and/or floor strata are weak in bearing strength, large bearing pressures can cause the chocks to penetrate into the strata and cause operational difficulties. These difficulties can be minimized by determining the bearing strength of the mine strata prior to installation of the support equipment. Support equipment considered safe and efficient for the conditions existing in the particular

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Both authors are with the Denver Mining Research Center, Bureau of Mines, Denver, Colo.

mine may then be selected such that the average bearing pressures exerted by the support will be less than the bearing strength of the strata.

The objective of this investigation was therefore (1) to develop a convenient, diagnostic, bearing capacity test and (2) to conduct sufficient tests in order to determine its effectiveness and reliability.

ACKNOWLEDGMENTS

J. S. Miller, formerly mining engineer, Bureau of Mines, assisted with the field testing. The assistance and cooperation received from officials and miners at the different mines is gratefully acknowledged.

STUDIES BY OTHER INVESTIGATORS

Various types of apparatus have been used to determine the inplace strength of coal measures and associated roof and floor strata. Principal among them are penetrometer tests (2, 5-6, 10)[3] and plate bearing tests (2-3, 6-8, 10). In addition, the civil engineering literature abounds with descriptions of methods to determine the bearing capacity of floors and foundations based on either a Soil Mechanics or a Theory of Elasticity interpretation (1, 9, 11-15).

With the usual penetrometer type of apparatus, a round rod of relatively small cross-sectional area, having either a conical, hemispherical, or flat end, is forced into the stratum being tested. The maximum resistance developed for a predetermined amount of penetration is measured. The equipment is usually very simple, portable, and designed for performing numerous tests over different areas with little difficulty. Test results show that, in general, the resistance of coal to penetration is as much as five times the uniaxial crushing strength of coal (3).

Later investigators (2-3, 6-8, 10) used a combination of the penetrometer and plate-load type of apparatus. Circular plates ranging in area from 0.014 in.^{2} (0.09 cm^2) to 108.5 in.^{2} (700 cm^2) were loaded until the stratum on which the plate was acting failed. Jenkins (7-8) plotted the rupture load (load at failure), R, against the plate diameter, D, and found a relationship $R = KD^n$ where K and n are, respectively, constants governed by the strength of the floor and the nature of loading between the bearing plate and the stratum tested. Jahns (6) plotted the specific breaking strength (load at failure divided by bearing plate area) against bearing plate area on semilog paper and produced a series of curves for different types of rocks (Rock Strength Standards). Both investigators then extrapolated the curves to predict the bearing capacity of rock when loaded by mine props.

Coates and Gyenge (3) employed plate loading to determine the Modulus of Deformation and the Bearing Capacity of the rock tested. Rigid, circular bearing plates, 4 to 8 in. in diameter (12.57 in.^{2} to 50.26 in.^{2} area), were

[3]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

used to test three types of rock. Results were not conclusive regarding the bearing capacity of the rock types, although a correlation was found between the modulus of deformation and the bearing capacity.

Dulaney (4) used rigid square bearing plates, ranging from 1½ to 12 inches square (2.25 in.^{2} to 144 in.^{2} area), to determine the bearing strength of mine floors in six different coal seams. The plates were placed on a ½-in.-thick layer of plaster of paris laid at each test location to minimize stress concentrations and to insure uniform loading.

None of the above-mentioned investigators extrapolated their results to determine the maximum allowable bearing pressure applied by footplates and canopies of longwall self-advancing supports.

The bearing pressure applied by the footplates and canopies of existing longwall self-advancing supports does not exceed 1,000 psi at the designed yield load value. The size of the footplates and canopies ranges from 12 by 12 in. to 36 by 108 in. and possibly larger.

From an engineering standpoint, the most desirable plate loading test is one that requires a minimum of extrapolation to the full size application. Consequently, the authors have used the largest possible plate sizes within the bounds of practicality in order to keep the extrapolation, if any, to a minimum.

DEVELOPMENT OF APPARATUS

Initially, the apparatus consisted of a 30-ton hydraulic prop, a 5,000-psi pressure gage and recorder, a 10,000-psi hand-operated hydraulic pump, dial gages to measure roof and floor penetration, prop extensions, and circular penetrometer plates of 1-, 2-, 3-, 4-, 5-, and 10-inch diameters.

Preliminary tests were made at two locations in a Colorado coal mine (mine 1). At the test locations, the floor consisted of weak-bedded shales and the roof consisted of at least 3 ft of coal overlain by shales. Most of the tests were made in the vicinity of an area being considered by mine management for longwall mining.

Table 1 gives the results of the tests. An attempt was made to keep the rate of loading constant during testing. The bearing pressure at which the strata initially failed was determined, as well as that for the failure, if any, of successive strata. Each test was concluded when a penetration of the plate into the strata amounted to 4 in. (a penetration of a support unit into the strata of greater than 4 in. would be undesirable from ground control considerations) or when the load capacity of the prop was reached (30 tons). Throughout this report the expression "bearing pressure" implies the average pressure over the area of the bearing plate, and is calculated from

$$\text{Bearing pressure} = \frac{\text{Applied load}}{\text{Area of bearing plate}}. \qquad (1)$$

TABLE 1. - Bearing capacity tests at mine 1, Colorado, circular plates

(Capacity of system = 30 tons)

Bearing plate diameter, in.	Bearing pressure, psi		
	At initial failure	At 2d failure	Maximum reached during test
TESTS ON FLOOR AT LOCATION A			
1	6,080	8,240	8,240
2	5,600	7,800	7,800
3	3,900	3,820	3,900
4	No failure	-	3,000
TESTS ON ROOF AT LOCATION B			
1	6,400	9,600	9,600
1	4,500	9,600	9,600
1	6,400	8,000	16,000
2	5,600	6,800	6,800
2	2,000	9,600	10,400
2	4,000	6,800	6,800
3	6,750	5,150	6,750
3	6,050	-	6,050
4	3,350	5,000	5,000
4	No failure	-	5,000
TESTS ON FLOOR AT LOCATION B			
1	1,300	-	3,200
1	1,600	-	2,840
1	4,050	-	4,320
2	1,600	-	1,600
2	2,400	-	2,400
2	4,200	-	4,200
3	2,130	-	2,130
4	1,240	-	1,240
4	800	1,500	1,500
4	700	-	700
5	830	-	830
5	1,090	-	1,090
5	1,130	-	1,130
10	No failure	-	800
10	640	-	640
10	750	-	750

Analysis of the results indicated that the apparatus had to be modified to accommodate larger penetrometer plates before realistic extrapolation of data to chock-size bearing areas could be done. The apparatus was accordingl modified and further tests were conducted at location B of the same mine. Th hand pump was replaced by an air-hydraulic pump so as to obtain better contro of the rate of loading. Also, larger circular plates of 8-, 10-, and 12-in. diameters were used in the tests. Table 2 gives the results obtained from these tests.

TABLE 2. - Bearing capacity tests on floor at location B, mine 1--circular plates

Bearing plate diameter, in.	Penetration rate, in./5 min.	Bearing pressure, psi	
		At initial failure	Maximum achieved
8	0.1	833	833
8	.1	598	598
8	.2	920	920
10	.2	1,122	1,122
10	.1	1,100	1,100
10	.1	661	661
12	.1	673	673
12	.1	414	414
12	.1	403	403

A total of 25 floor tests were thus made with circular bearing plates at location B in mine 1. The results of these tests are summarized in table 3. The relationship (fig. 1) between the maximum bearing pressure imposed during 4 in. of penetration and the area of the circular penetration plate was computed to be

$$P_{max} = 3{,}311 \text{ (bearing area)}^{-0.36} \quad (2)$$

or

$$\text{Failure load} = 2.837D^{1.28}, \quad (3)$$

where D is the diameter of the plate, and P_{max} = maximum bearing pressure = failure load/bearing area.

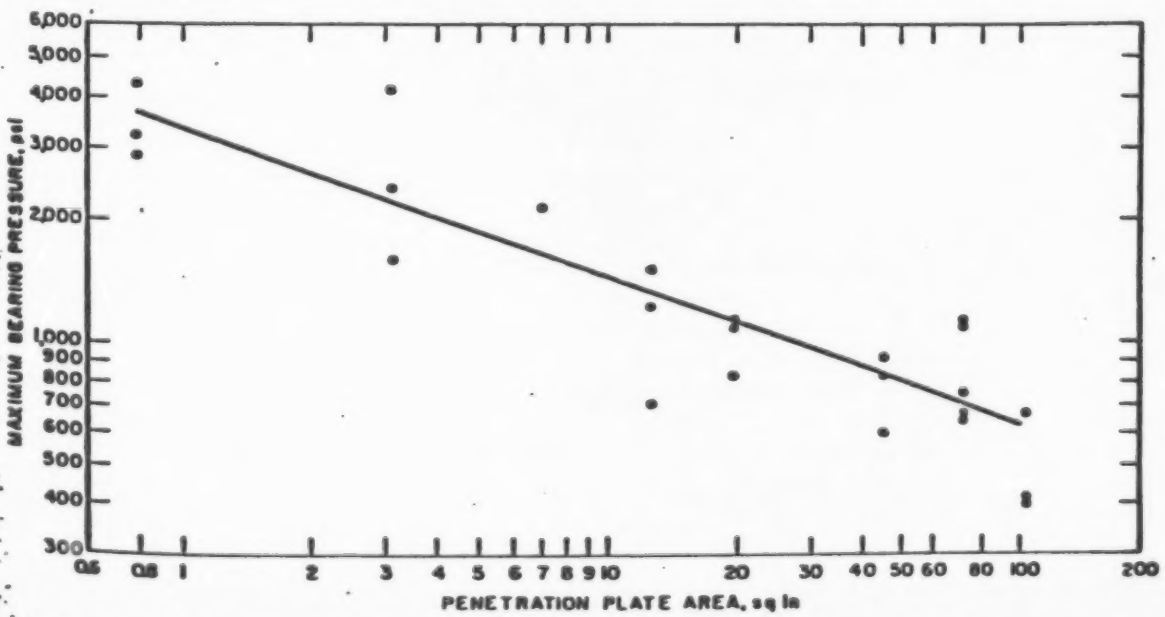


FIGURE 1. - Relationship Between Maximum Bearing Pressure During Total Penetration and Areas of Circular Penetration Plates, Mine 1, Colorado.

TABLE 3. - Summary of bearing capacity tests on floor at location B, mine 1, Colorado--circular plates

Test No.	Plate diameter, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate (during test), tons	Bearing pressure at initial failure, psi	Maximum bearing pressure achieved, psi
CONSTANT RATE OF LOADING; SYSTEM CAPACITY = 30 TONS						
1	1	0.75	4.70	1.1	1,280	3,200
2	1	.5	3.66	1.25	1,920	2,880
3	1	1.5	3.78	1.7	4,050	4,320
4	2	2.5	2.20	2.5	1,600	1,600
5	2	3.8	4.72	3.8	2,400	2,400
6	2	6.6	4.34	6.6	4,200	4,200
7	3	7.5	4.38	7.5	2,130	2,130
8	4	7.8	4.08	7.8	1,240	1,240
9	4	5.0	3.94	9.4	800	1,500
10	4	4.4	3.60	4.4	700	700
11	5	8.2	3.98	8.2	830	830
12	5	6.3	4.08	10.6	640	1,090
13	5	11.1	4.05	11.1	1,130	1,130
14	10	No failure	.95	31.4	No failure	800
15	10	25.75	4.06	25.75	640	640
16	10	28.5	3.46	28.50	750	750
CONSTANT RATE OF PENETRATION; SYSTEM CAPACITY = 100 TONS						
17	8	18	4.08	18	833	833
18	8	13.4	4.00	13.4	598	598
19	8	20.6	4.01	20.6	920	920
20	10	39.7	4.40	39.7	1,100	1,100
21	10	23.9	4.00	23.9	661	661
22	10	40.5	4.04	40.5	1,122	1,122
23	12	34.9	3.93	34.9	673	673
24	12	21.5	4.00	21.5	414	414
25	12	20.9	4.03	20.9	403	403

Equations 2 and 3 can be written in the general form

$$P = K'A^{-n} \quad (4)$$

and

$$R = KD^{m} \quad (5)$$

where P = bearing pressure at failure,

R = rupture or failure load,

A = area of bearing plate, and

m, n, K, K' = constants.

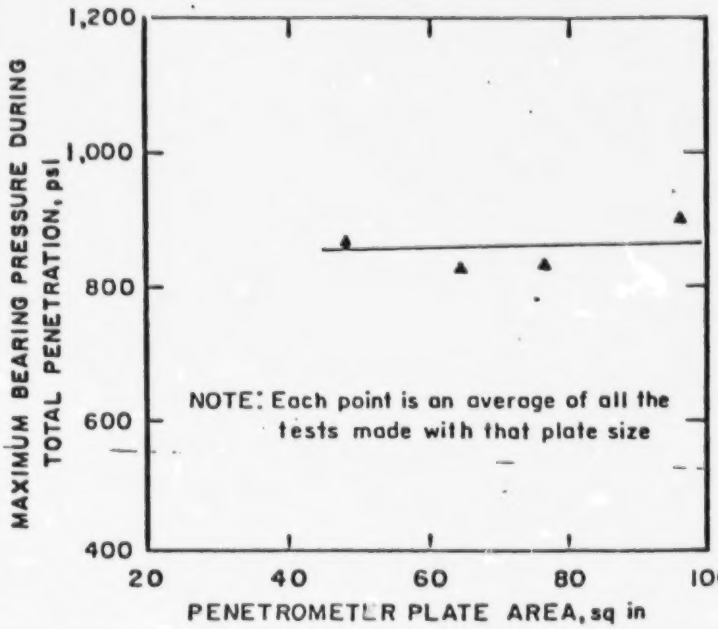


FIGURE 2. - Results of Bearing Capacity Tests at Colorado Mine With Rectangular Plates.

Equations 4 and 5 have been shown to hold true for different types of roof and floor strata in Europe by Jenkins (7-8) and Jahns (6).

There are extreme differences between the shapes and sizes of the circular bearing plates used in bearing capacity tests and the rectangular footplates and canopies of self-advancing longwall supports. Precautions must be used in extrapolating relationships similar to equations 2 and 3 to obtain the maximum allowable bearing pressure to be applied by the footplates and canopies.

It was therefore decided to reduce the differences between the size and shape of the bearing plate used in the tests and the prototype, by using square and rectangular bearing plates of sizes ranging from 6 by 6 in. to 12 by 12 in. and 6 by 18 in.

Results of tests made on the floor at location B in mine 1 with the rectangular bearing plates are given in table 4. Figure 2 is a plot of the average bearing capacity of the floor for a given plate versus the plate area.

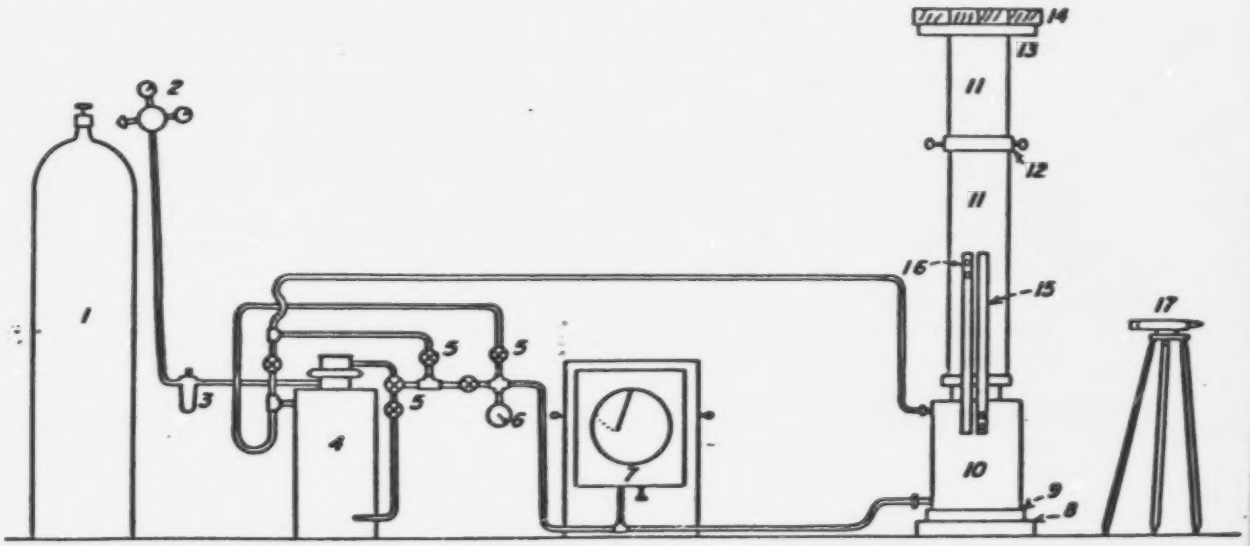
TABLE 4. - Bearing capacity tests on floor at mine 1, Colorado--rectangular plates

Test No.	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure reached, psi	
						Individual test	Plate size average
1	6 x 12	27.7	3.97	27.7	1,157	1,157	-
2	6 x 12	18.5	3.92	18.5	775	775	864
3	6 x 12	14.8	4.00	14.8	619	661	-
4	6 x 18	24.5	4.00	24.5	638	638	-
5	6 x 18	39.5	4.00	39.5	1,031	1,031	835
6	6 x 18	32.0	4.00	32.0	835	835	-
7	8 x 12	38.0	3.98	38.0	1,180	1,180	-
8	8 x 12	19.3	4.00	19.3	600	600	824
9	8 x 12	22.3	4.02	22.3	692	692	-
10	8 x 16	54.0	4.00	54.0	1,113	1,113	-
11	8 x 16	53.5	4.00	53.5	1,113	1,113	908
12	8 x 16	23.6	4.00	23.6	452	497	-

In spite of the considerable scatter of individual tests, the plot indicates that the average bearing capacity per plate does not vary significantly with the size of the bearing plate. This differs from the relationships (equations 4 and 5)determined from tests with circular plates. It appears that an estimate of the average bearing capacity that is independent of the size of the bearing plate may therefore be obtained by averaging the bearing capacity values from tests made with as many of the rectangular plates as desired. The test was therefore modified to utilize only square and rectangular plates. The number of tests made to obtain reliable estimates of the lower limit of the bearing capacity was determined from the first three tests. If the average bearing capacity as determined by the first three tests was greater than 6,000 psi (capacity of the system using a 6- by 6-in. plate), no further tests were made. With lower average bearing capacity, correspondingly larger number of tests were made. Since the main purpose was to obtain a simple and efficient method of testing the bearing capacity of mine strata, the maximum number of tests made on any particular strata in a particular location was limited to a total of nine, made up of three tests each with three different plate sizes.

DESCRIPTION OF APPARATUS

The apparatus in its final form is shown schematically in figure 3 and includes:



1 Nitrogen cylinder
2 Air pressure regulator connected to compressed air or N_2 cylinder
3 Air-line lubricator
4 1:150 air-hydraulic pump
5 High-pressure valves
6 20,000-psi pressure gage
7 10,000-psi pressure recorder
8 Penetration plate
9 Adapter
10 100-ton ram
11 Tubular steel extension
12 Bolted sleeve
13 Bearing plate 14in x 14in
14 Hardwood blocking
15 Floor penetration scale
16 Roof penetration scale
17 Sighting telescope

FIGURE 3. - Schematic of Bearing Capacity Test Apparatus.

a. A 100-ton-capacity hydraulic ram with a stroke of about 10 in.

b. An air-hydraulic pump with an air to fluid pressure ratio of 1 to 150, capable of delivering fluid pressures beyond the ram capacity and capable of maintaining constant fluid pressure for selected periods of time.

c. A 0- to 10,000-psi clock-wound continuous pressure recorder with a chart rotation time of 4 hours.

d. A level telescope or equivalent sighting device to measure strata penetration and ram extension remotely by observing movement of separately attached scales having 0.02-in. divisions.

e. A nitrogen cylinder (2,000 psi) or compressed-air supply.

f. An air regulator capable of controlling air pressure from 0 to 100 psi.

g. A set of rectangular penetrometer plates of dimensions as shown in figure 4 and table 5. The plates are of 2-in. thickness to minimize distortion under load. Edges of the plates are rounded to minimize stress concentration.

h. A 14- by 14-in. steel bearing plate with accessory hardwood blocking is used such that a bearing area greater than the penetrometer plate being used is provided at the end opposite to the penetrometer plate.

i. Miscellaneous parts, including high-strength steel tubular sectionalized extensions, high-pressure hydraulic lines, high-pressure valves and fittings, and construction tools required for preparation of test sites.

Figure 5 is a view of the apparatus prior to the bearing capacity test in a mine.

TEST PROCEDURE

The following is a description of the final procedure adopted to assure uniformity of all bearing capacity tests.

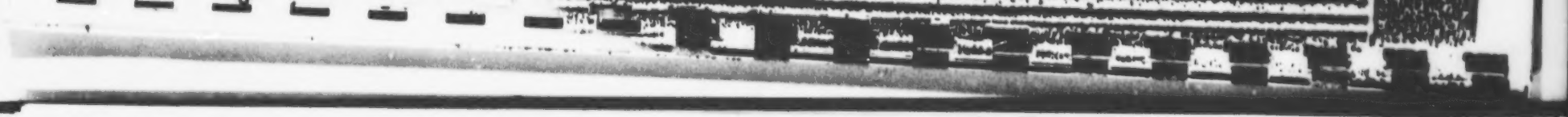
Selection of Test Sites

Each test site was in a development entry in that portion of the mine where a longwall was operating or where a longwalling operation was being considered. The test sites appeared to be representative of the conditions normally present in the mine. This precluded tests under the following conditions:

a. Abnormal quantities of water.

b. Fractured bottom or top.

c. Local anomalies such as faulted areas and pinch-outs.



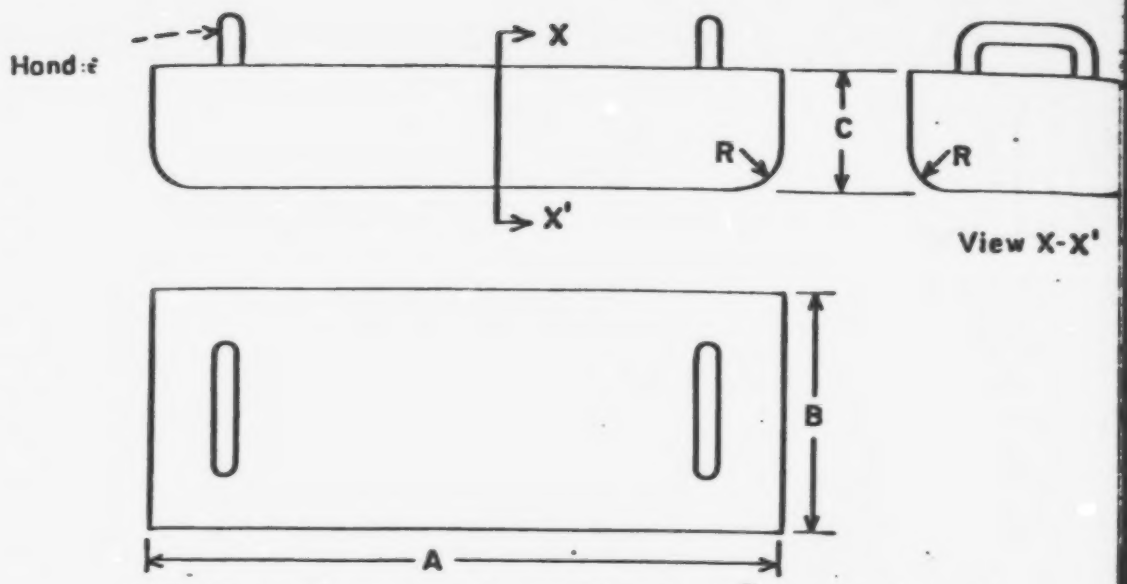


FIGURE 4. - Penetrometer Bearing Plate Dimensions. (See table 5.)

TABLE 5. - Penetrometer bearing plate dimensions

Plate No.	Dimension, in.				Area, in.^{2}
	A	B	C	R	
1	6	6	2	3/8	31.64
2	9	6	2	3/4	34.73
3	12	6	2	3/4	47.81
4	18	6	2	3/4	76.60
5	8	8	2	3/8	56.72
6	12	8	2	3/4	64.44
7	16	8	2	3/4	96.06
8	10	10	2	3/8	70.28
9	12	12	2	3/8	132.25

Information regarding the character and thicknesses of the immediate r and floor strata was obtained. Occasionally, it was necessary to hand dril the roof and floor to obtain this information.

Preparations for Test

The floor and roof were cleaned of loose material, and the test surfac were made flat to achieve reasonably uniform loading. The bearing area of assembly at the end opposite the penetrometer plate was made much larger th the penetrometer plate through the use of the large steel bearing plate and hardwood blocks. Wooden wedges were used to provide full contact of the blocking with the roof.

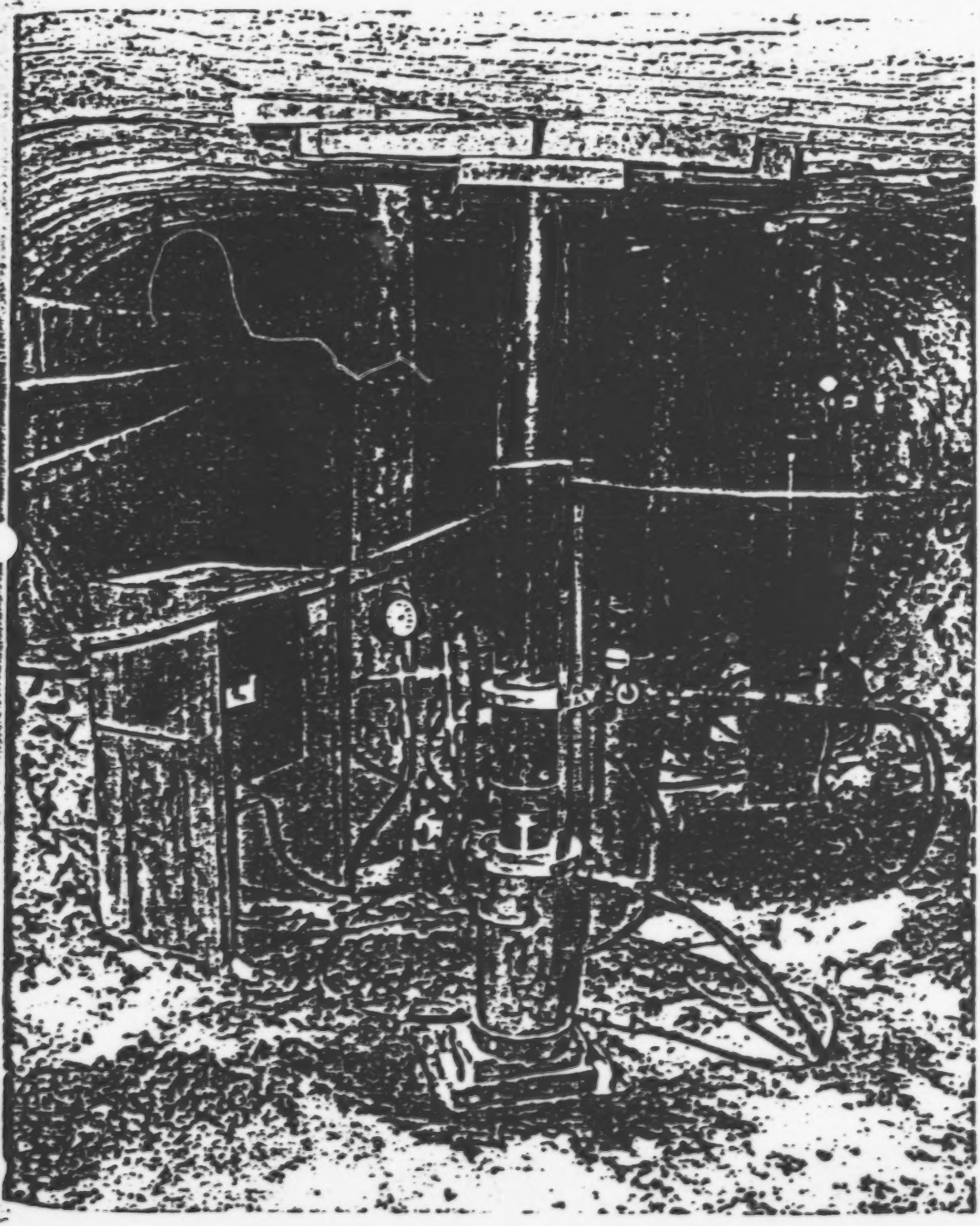


FIGURE 5. - Apparatus Used To Test the Bearing Capacity of Mine Roof and Floor.

MERGENTHALER LINOTYPE CO.
BROOKLYN, NEW YORK

The graduated scales were attached separately to the ram and the casing so as to permit measurement of penetration of the bearing plate into the strata and deformation of the hardwood blocking. The level telescope was set up about 10 ft away from the assembly so that it was outside the influence of possible strata movements caused by the test. It was sighted on the two graduated scales. The line of sight through the telescope served as an independent datum against which the movements of the scales were noted.

Procedure

The system was assembled as shown in figure 3. The load on the selected penetrometer plate was increased by operation of the air-hydraulic pump. The rate of loading was controlled by manipulation of the air-line regulator such that a constant rate of penetration of 0.2 in. per 5-minute interval was maintained. The tests were terminated when either a 4-in. penetration of the plate into the strata or the load capacity of the ram (100 tons) was reached. If the strata failed, the load at initial failure was noted. Failure of the strata was determined by that bearing pressure between the plate and the rock at which the penetration rate increased noticeably while being accompanied by a drop in the recorded fluid pressure. It was indirectly determined by the maximum pressure in the recorded plot of fluid pressure versus time. A

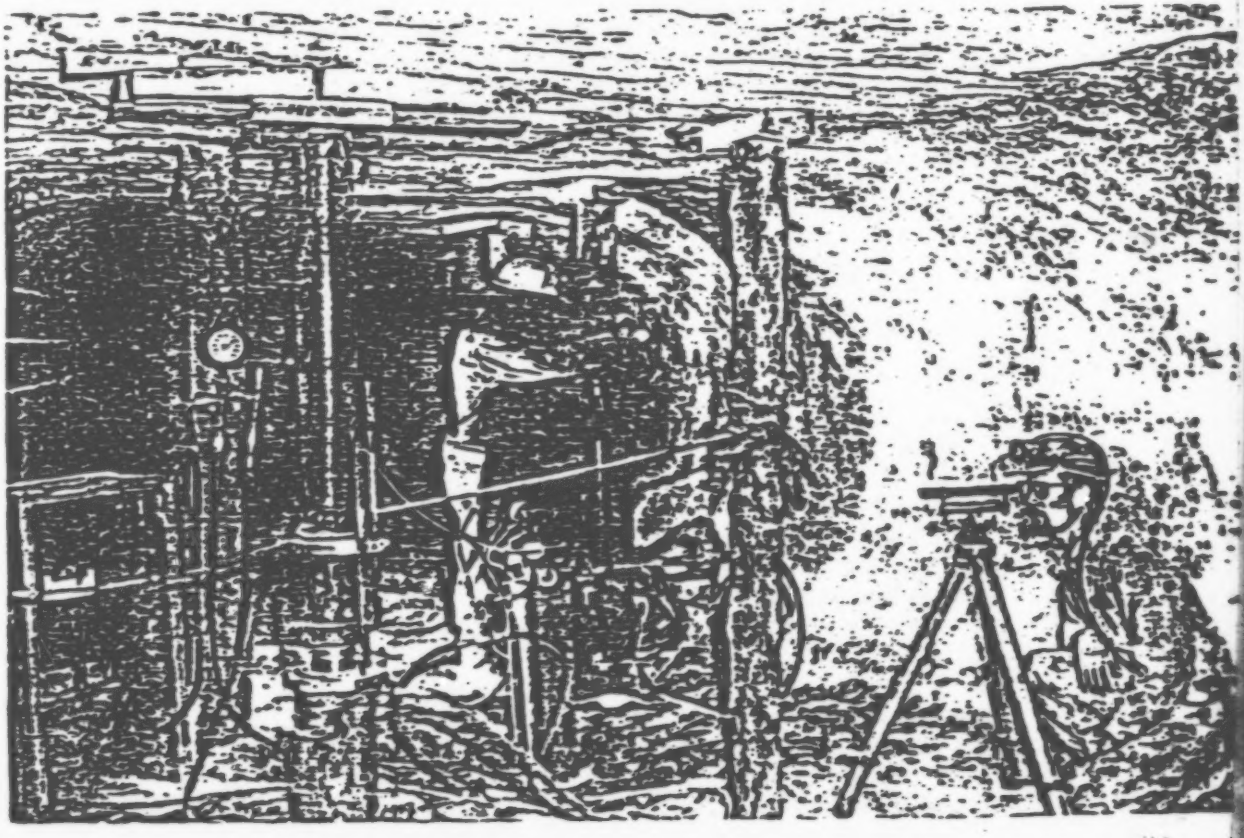


FIGURE 6. - Bearing Capacity Test in Progress at Mine I.

onstant rate of penetration was maintained even after failure. In cases here layers of the strata failed independently, bearing pressures at successive failures of the strata were also noted.

The penetrometer plate selected for the first test was usually of the largest size that could reasonably be expected to produce failure at loads of less than 100 tons. The sequence of plate sizes used in successive tests were randomized, and the spacing between test points was always more than 5 ft.

Figure 6 shows the bearing capacity test apparatus being used to test the floor in a coal mine.

DESCRIPTION OF FIELD TESTS AND ANALYSIS OF RESULTS

Tests of the roof and floor were made at eight other mines, located in seven States, selected as being representative of coal measures and of operating longwalls in the United States. The results are given in table 6 and are summarized in table 7. Figure 7 gives the stratigraphy of the immediate roof and floor in the test areas of each of the mines where bearing capacity tests were made.

In mines where the roof and/or floor strata were strong, the strata did not fail at the capacity of the system (100 tons) even when the smallest bearing plate (6 by 6 in.) was used, hence the average bearing capacity exceeds 6,000 psi, the maximum bearing pressure applied by our system. The roofs of mines 2, 3, 4, 7, and 9 and the floors of mines 2 and 4 fall in this category.

In other mines, as in the floors of mine 3, mine 8, and mine 9 and the roof of mine 5, the results indicate that the bearing capacity depends upon the strength of the immediate layer tested and varies, for example, with the thickness of the coal in the floor or the thickness of draw slate in the immediate roof. In mines with similar strata, it is suggested that sufficient tests be made to obtain reliable estimates of bearing capacity for the different layers of the roof and/or floor strata. It may then be possible to determine, for example, the optimum thickness of floor coal to be left such that the floor remains competent under operating conditions.

Tests made at mine 8 indicate that the bearing capacity of the fireclay layer, beneath the 4- to 8-in. thickness of shale that comprises the floor of the longwall panel, is much greater than that of the shale layer. If the shale proved to be incompetent and allowed penetration of the longwall supports into the floor under operating conditions, it might prove to be advantageous from a ground control standpoint to include the shale layer in the cut made by the mining machine and to use the fireclay as a competent floor. Results of tests made at mine 9 show a similar situation, with indications being that the shale layer below the fireclay and coal in the floor is the strongest of the three layers. However, in this mine, the longwall floor has a 4-in. layer of hard pan over the 4-in. layer of coal and apparently this hard pan layer contributes to a competent floor.

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TABLE 6. - Bearing capacity tests, [illegible]

Test	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure, psi	Remarks
MINE 2, UTAH							
Floor:							
1	6 x 9	No failure	0.07	107.5	>6,205	6,205	Shale floor.
2	6 x 6	do.	.10	105.5	>6,675	6,675	Do.
3	6 x 6	do.	.20	106.5	>6,743	6,743	Do.
4	6 x 6	do.	.10	104.5	>6,607	6,607	Do.
Roof:							
1	6 x 6	do.	.42	107.5	>6,810	6,810	Shale roof.
2	6 x 6	do.	.36	107.5	>6,810	6,810	Do.
3	6 x 6	do.	.36	107.5	>6,810	6,810	Do.
MINE 3, ILLINOIS							
Floor:							
1	6 x 6	89.5	2.51	89.5	5,660	5,660	Coal floor; 20 in. of coal.
2	8 x 8	84	4.10	84	2,970	2,970	14 in. of coal.
3	8 x 8	No failure	.50	103	>3,630	3,630	Over 24 in. of coal.
4	6 x 9	do.	.58	101.5	>5,850	5,850	Do.
5	10 x 10	do.	.63	104	>2,870	2,870	Do.
6	6 x 6	do.	.60	105	>6,640	6,640	Do.
7	6 x 6	78	2.48	82.5	4,930	5,280	Do.
8	6 x 9	No failure	.29	104.5	>6,020	6,020	Do.
Roof:							
1	6 x 6	84	3.54	84	5,320	5,320	Shale roof.
2	6 x 9	No failure	.41	101.5	>5,860	5,860	Do.
3	6 x 6	do.	.47	103	>6,500	6,500	Do.
4	6 x 9	do.	.33	104	>6,000	6,000	Do.
MINE 4, VIRGINIA							
Floor:							
1	6 x 6	81	3.95	81	5,145	5,145	Hard fireclay floor.
2	8 x 8	No failure	.20	108	>3,800	3,800	Do.
3	6 x 9	do.	.19	110	>6,340	6,340	Do.
4	6 x 6	do.	.10	111	>7,000	7,000	Do.
5	6 x 6	do.	.10	109	>6,950	6,950	Do.
Roof:							
6	6 x 6	do.	.34	108	>6,800	6,800	Shale roof.
7	6 x 6	do.	1.03	110	>6,975	6,975	Do.
8	6 x 6	do.	1.09	109	>6,900	6,900	Do.
MINE 5, WEST VIRGINIA							
Floor:							
9	6 x 6	38	4.00	55	2,390	3,480	Shale/fireclay floor.
10	8 x 8	No failure	.65	109	>3,855	3,855	Do.
11	6 x 6	60	3.59	60	3,786	3,786	Do.
12	6 x 9	62	3.94	62	3,580	3,580	Do.
13	6 x 9	59	3.99	59	3,430	3,430	Do.
14	6 x 6	26	3.87	26	1,612	1,612	Do.
15	8 x 8	104	1.86	104	3,666	3,666	Do.
16	6 x 9	82	2.51	82	4,720	4,720	Do.
17	8 x 8	79	3.36	79	2,780	2,780	Do.
Roof:							
1	6 x 6	45	4.00	45	2,835	2,835	20 in. of draw slate.
2	10 x 10	No failure	.52	109	>3,070	3,070	30 in. of draw slate.
3	6 x 18	do.	.37	109	>2,840	2,840	Do.
4	6 x 12	do.	.30	107	>4,500	4,500	28 in. of draw slate.

TABLE 6. - Bearing capacity tests, [illegible]

Test	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure, psi	Remarks
MINE 2, UTAH							
Floor:							
1	6 x 9	No failure	0.07	107.5	>6,205	6,205	Shale floor.
2	6 x 6	do.	.10	105.5	>6,675	6,675	Do.
3	6 x 6	do.	.20	106.5	>6,743	6,743	Do.
4	6 x 6	do.	.10	104.5	>6,607	6,607	Do.
Roof:							
1	6 x 6	do.	.42	107.5	>6,810	6,810	Shale roof.
2	6 x 6	do.	.36	107.5	>6,810	6,810	Do.
3	6 x 6	do.	.36	107.5	>6.810	6,810	Do.
MINE 3, ILLINOIS							
Floor:							
1	6 x 6	89.5	2.51	89.5	5,660	5,660	Coal floor; 20 in. of coal.
2	8 x 8	84	4.10	84	2,970	2,970	14 in. of coal.
3	8 x 8	No failure	.50	103	>3,630	3,630	Over 24 in. of coal.
4	6 x 9	do.	.58	101.5	>5,850	5,850	Do.
5	10 x 10	do.	.63	104	>2,870	2,870	Do.
6	6 x 6	do.	.60	105	>6,640	6,640	Do.
7	6 x 6	78	2.48	82.5	4,930	5,280	Do.
8	6 x 9	No failure	.29	104.5	>6,020	6,020	Do.
Roof:							
1	6 x 6	84	3.54	84	5,320	5,320	Shale roof.
2	6 x 9	No failure	.41	101.5	>5,860	5,860	Do.
3	6 x 6	do.	.47	103	>6,500	6,500	Do.
4	6 x 9	do.	.33	104	>6,000	6,000	Do.
MINE 4, VIRGINIA							
Floor:							
1	6 x 6	81	3.95	81	5,145	5,145	Hard fireclay floor.
2	8 x 8	No failure	.20	108	>3,800	3,800	Do.
3	6 x 9	do.	.19	110	>6,340	6,340	Do.
4	6 x 6	do.	.10	111	>7,000	7,000	Do.
5	6 x 6	do.	.10	109	>6,950	6,950	Do.
Roof:							
6	6 x 6	do.	.34	108	>6,800	6,800	Shale roof.
7	6 x 6	do.	1.03	110	>6,975	6,975	Do.
8	6 x 6	do.	1.09	109	>6,900	6,900	Do.
MINE 5, WEST VIRGINIA							
Floor:							
9	6 x 6	38	4.00	55	2,390	3,480	Shale/fireclay floor.
10	8 x 8	No failure	.65	109	>3,855	3,855	Do.
11	6 x 6	60	3.59	60	3,786	3,786	Do.
12	6 x 9	62	3.94	62	3,580	3,580	Do.
13	6 x 9	59	3.99	59	3,430	3,430	Do.
14	6 x 6	26	3.87	26	1,612	1,612	Do.
15	8 x 8	104	1.86	104	3,666	3,666	Do.
16	6 x 9	82	2.51	82	4,720	4,720	Do.
17	8 x 8	79	3.36	79	2,780	2,780	Do.
Roof:							
1	6 x 6	45	4.00	45	2,835	2,835	20 in. of draw slate.
2	10 x 10	No failure	.52	109	>3,070	3,070	30 in. of draw slate.
3	6 x 18	do.	.37	109	>2,840	2,840	Do.
4	6 x 12	do.	.30	107	>4,500	4,500	28 in. of draw slate.

TABLE 6. - Bearing capacity tests, mines 2-9--Continued

Test	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure, psi	Remarks
MINE 5, WEST VIRGINIA--Continued							
Roof: (Con.)							
5	6 x 6	No failure	0.46	104	>6,600	6,600	30 in. of draw slate.
6	6 x 6	do.	.45	104	>6,600	6,600	Do.
7	6 x 6	60	4.00	70	3,810	4,398	Do.
8	6 x 6	No failure	.37	107	>6,776	6,776	Do.
MINE 6, WEST VIRGINIA							
Floor:							
4	6 x 6	88	3.48	88	5,552	5,552	Shale floor.
5	8 x 8	79	3.50	79	2,794	2,794	Do.
6	8 x 8	88	3.94	88	3,115	3,115	Do.
7	8 x 8	77	4.00	77	2,733	2,733	Do.
8	6 x 6	42	4.00	42	2,632	2,632	Do.
9	6 x 6	63	4.02	63	3,990	3,990	Do.
Roof:							
1	6 x 6	67	3.94	72	4,260	4,570	Shale roof.
2	6 x 6	No failure	.32	103	>6,530	6,530	Do.
3	6 x 6	do.	.19	105	>6,640	6,640	Do.
MINE 7, WEST VIRGINIA							
Floor:							
4	6 x 6	54	3.74	54	3,447	3,447	On shale.
5	8 x 8	No failure	2.16	104	>3,680	>3,680	Verge of failure.
6	8 x 8	102	3.34	102	3,590	3,590	On shale.
7	6 x 6	45	4.12	45	2,835	2,835	Do.
8	8 x 8	99	4.12	99	3,490	3,490	Do.
9	6 x 6	57	4.10	57	3,610	3,610	Do.
Roof:							
1	6 x 6	No failure	.84	104	>6,572	6,572	On coal top.
2	6 x 6	do.	1.34	105	>6,670	6,670	Do.
3	6 x 6	do.	.51	104	>6,600	6,600	Do.
MINE 8, PENNSYLVANIA							
Floor:							
1	6 x 6	62	4.00	62	3,922	3,922	Shale floor.
2	8 x 8	49	3.15	49	3,107	3,107	2 in. of floor coal.
3	6 x 6	No failure	.36	97	>6,164	6,164	Hard fireclay floor.
4	6 x 6	do.	.54	102	>6,436	6,436	Do.
8	6 x 6	do.	.64	102	>6,436	6,436	Do.
Roof:							
5	6 x 6	do.	.28	95	>6,028	6,028	4 in. of top coal.
6	6 x 6	79	4.00	79	5,009	5,009	Do.
7	6 x 6	No failure	.20	102	>6,436	6,436	Sandy shale roof.
MINE 9, PENNSYLVANIA							
Floor:							
1	6 x 6	40	3.19	40	2,563	2,563	Coal floor.
2	8 x 8	77	4.00	77	2,730	2,730	Do.
3	8 x 8	44	3.94	44	2,800	2,800	Do.
4	6 x 6	32	3.90	37	2,020	2,360	Do.
5	8 x 8	60	3.16	107	2,112	3,780	Fireclay floor.
6	6 x 6	81	4.10	81	5,145	5,145	Shale floor.
Roof:							
7	6 x 6	No failure	.23	98	>6,232	6,232	Shale roof.
8	6 x 6	do.	.17	103	>6,504	6,504	Do.
9	6 x 6	do.	.80	103	>6,530	6,530	Do.

TABLE 7. - Summary of bearing capacity test results

Mine No.	1	2	3	4	5[1]
Area	Colorado	Utah	Illinois	Virginia	West Virginia
Cover, ft	350	1,000-2,000	660	1,350-2,000	500-800
Seam thickness, in.	108-144	65-80	108	52	50
Height of extraction, in.	84	82	84	52	50
Test location	4 S off main W	16 left	3 N	#4 development entry-#2 plow	1st right
Roof geology	Coal	Sandy shale	Shale	Sandstone over 18 to 72 in. shale	18 to 60 in. draw slate up to 4 in. rider seam
Roof, average bearing capacity, psi	>3,000	>6,000	>5,000	>6,000	>5,000
Floor geology	Shale	Shale	2 ft coal over fireclay	Hard fireclay	6 in. shale over fireclay
Floor, average bearing capacity, psi	800	>6,000	>5,000	>6,000	3,400

Mine No.	6	7[2]	8	9[3]
Area	West Virginia	West Virginia	Pennsylvania	Pennsylvania
Cover, ft	400-800	400-600	440-600	500-800
Seam thickness, in.	60	80	58	54
Height of extraction, in.	60	72	54	50
Test location	11 butt left	6 right off 7 N mains	Face area	D west
Roof geology	Shale	Draw slate over 8 in. top coal	4 in. top coal, then sandy shales	Shale
Roof, average bearing capacity, psi	>4,500	>6,500	>5,000	>6,000
Floor geology	Shale	Shale	6 in. shale over 24 in. fireclay over sandstone	6 in. coal over 4 in. fireclay over shale
Floor, average bearing capacity, psi	3,400	3,400	Shale floor = 3,500; fireclay floor >6,000	Coal floor = 2,700; shale floor >3,800

[1]Strength of roof varies directly as thickness of draw slate.
[2]6 in. bottom coal left on longwall floor.
[3]Longwall floor has 6 in. layer of hard pan over coal.

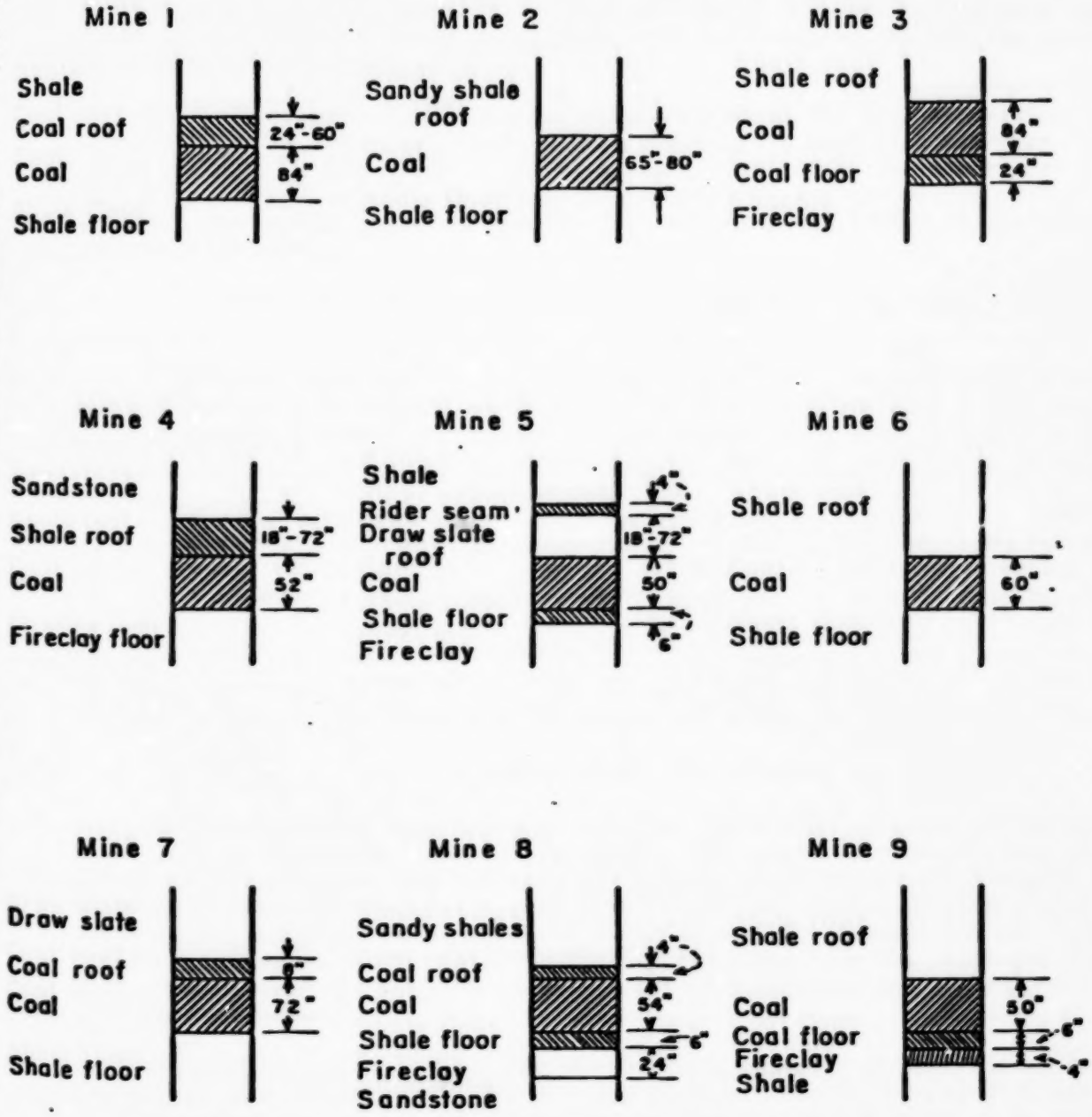

FIGURE 7. - Stratigraphy of Immediate Roof and Floor in Test Areas.

Mines 2, 4, 5, 7, 8, and 9 had operating longwalls at the time the tests were conducted. In each of these mines the estimates of the lower limit of the bearing capacity of the roof and the floor as determined by our tests were compared with the maximum bearing pressure applied by the hydraulic support on the longwall as obtained from the manufacturer's specifications. In each case the estimate of the lower limit of the bearing capacity was found to be much greater than the maximum bearing pressure of the support. None of the

longwalls had any unusual difficulty with supports punching into the floor, which indicates that the lower limit of bearing capacity of mine strata as estimated by the bearing tests may be used to determine the maximum allowable bearing pressure for a support. Additional tests conducted at mines that have poor floor/roof will provide complete proof of the reliability of this bearing capacity test. Future tests are being considered in mines having roof and floor of questionable bearing capacity with reference to the bearing pressures applied by longwall supports.

In analyzing the data obtained from testing in the nine mines, the lowest bearing capacity value as obtained from the test in each mine has been taken as an estimate of the lower limit of bearing capacity of the floor/roof in that mine. These values are tabulated in table 8; for comparison we have also included the design value of bearing pressure at yield load, for the support used in each mine.

TABLE 8. - Bearing capacity: Average and lower limits

Mine	Floor			Roof		
	Average bearing capacity, psi	Estimate of lower limit, psi	Design bearing pressure of support in use at yield, psi	Average bearing capacity, psi	Estimate of lower limit, psi	Design bearing pressure of support in use at yield, psi
1	858	490	No longwall	-	-	-
2	>6,000	>5,000	400	>6,000	>5,000	200
3	>4,820	2,970	No longwall	>5,000	>5,000	No longwall
4	>5,000	>4,000	780	>6,000	>5,000	100
5	3,420	1,600	No longwall	>4,600	>2,800	No longwall
6	3,470	2,600	780	>5,000	>4,000	100
7	3,440	2,800	800	>6,000	>5000	500
8	5,000	3,100	510	>5,000	>4,000	360
9	2,890	2,000	200	>6,000	>5,000	100

DISCUSSION AND CONCLUSIONS

Results obtained from bearing capacity tests with circular plates agree with prior work done by Jenkins and Jahns. The accuracy of estimating allowable bearing pressures for rectangular longwall base plates and canopies is improved by carrying out tests with square and rectangular bearing plates of large areas, in order to minimize extrapolation. The average and the lower limit of bearing capacity of the strata as obtained from the tests may then be used to determine the maximum allowable bearing pressure for a chock of a specified yield load.

The tests in mine 1 show that the bearing capacity of rock varies according to the size and shape of the bearing plates. The tests also indicate that the rate of change of bearing capacity decreases as the bearing plate diameter is increased. The average bearing capacity of a rock layer tends to become a constant when tests are made with plates of large areas.

Owing to the natural inhomogeneity of strata found in the mining environment, the scatter of the results obtained from the bearing capacity tests is rather pronounced. It is evident that the results obtained are a function of the strength of the strata and also of the test procedure. We have attempted to minimize the latter variable by following a standard test procedure. The average bearing capacity obtained from our tests is thus an estimate of the true average bearing strength of the rock.

The estimate of the lower limit of bearing capacity as determined by the lowest value of the bearing capacity tests made on the roof/floor of a mine may be used to determine the maximum allowable bearing pressure that may be applied by a longwall support on the roof/floor of that mine. Before making a final decision about introducing a longwall support system, bearing tests should be made to determine the capacity of the strata to support load over a period of several days. Such tests would simulate conditions such as the standing of a wall over a weekend.

The bearing strength of the strata may be considerably weakened by water being made in the longwall area and it may be considerably strengthened through selective mining, such as leaving a 12-in.-thick layer of top or floor coal. For proper interpretation of the tests, the tests should be made under essentially similar conditions to those under which the longwall will be operated.

Most of the operators of longwalls in the United States appear to have a good idea of the strength of the roof and floor strata with respect to the bearing pressures applied by their support equipment. This knowledge has come through experience and trial and error. The merits of doing bearing capacity tests lie in that knowledge of the strength of the rock is obtained before selecting and installing longwall equipment, which will help to avoid costly mistakes and delays for both the operator and the equipment manufacturer.

The test procedure and the test results presented herein show that a convenient, effective, and reliable test to determine the bearing capacity of mine roof and/or floor has been developed. The apparatus can be made quite easily by any mining company; the tests can be made by untrained personnel working under some engineering supervision; the results of the tests may be used without any questionable extrapolation, and the expense involved is minimal.

Determination of the bearing capacity of the mine strata in the vicinity of a proposed longwall prior to selection of support equipment may be well worth the effort when the dangers, the difficulties, and the expenses that occur due to the selection of inappropriate support systems are considered.

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4. Punch

Chlumecky (1968) described a method for testing the bearing capacity of floor strata. He incorporated standard plate bearing tests to determine the required bearing area of the supports. Barry and Nair (1970) also reported on the bearing capacity of roof and floor strata in coal mines. Their results were similar to those of Jahns. For large diameter circular bearing plates, the maximum allowable bearing pressure at failure becomes:

$$\text{Failure Load} = 2.837\ D^{1.28}$$

where D = diameter of the plate

To determine the maximum allowable support bearing pressure for longwalls, they conducted plate bearing tests with square and rectangular plates of areas ranging ranging from 36 $in.^2$ to 108 $in.^2$, took the lowest value, and set this for the maximum permissable pressure. The results of this research are included with program.

To determine the tendency of the support to punch requires this area of the base and canopy to be specified. In reality, the unequal loading on the base with the highest stress concentration on the foward edge might punch before the entire unit. Therefore, as this has not been studied or reported in the literature, a safety factor of 3.0 is used in the program.

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